Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET

Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

PROCEEDINGS OF THE MEETING OF THE CANADIAN URANIUM PRODUCERS' METALLURGICAL COMMITTEE HELD IN SASKATOON, MAY 19-20, 1981

G.M. RITCEY
Extractive Metallurgy Laboratory

April 1982

Project 4.4.0.2.01
Uranium Extraction - Conventional Technology

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MICROMEDIA

ENERGY RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 82-42(TR)

PREFACE

This publication has been prepared to provide a permanent record of the Proceedings of the 1981 Annual Meeting of the Canadian Uranium Producers' Metallurgical Committee which was held in Saskatoon, Saskatchewan. The author's original unedited technical presentations together with the discussions have been incorporated into this report.

The next annual meeting of the Committee is scheduled for May 1982 and will be held in Bancroft, Ontario.

ACKNOWLEDGEMENTS

The participation of those who attented is to be commended and is the main factor that demonstrates the usefulness of the meeting. The generous hospitality provided by Gulf Minerals and Eldorado Nuclear during the post meeting tours, and the tour of Saskatchewan Research Council certainly added to the success of the meeting. Acknowledgement is also made to Mrs. L. Paquette, CANMET, who assisted in the preparation of this record of the meeting.

G.M. Ritcey
Secretary, CUPMC

BUSINESS SESSION

A short business meeting concluded the two day meeting. The bank balance was stated as $19.45 as of April 30, 1981.

The next meeting of the CUPMC was set for May 1982 in Bancroft, Ontario, with Madawaska the host, and Leo McFarlane as Chairman.

Gulf Minerals was thanked for providing the arrangements of the meeting and the tour of the Rabbit Lake Mill; Eldorado Nuclear for the tour of the Beaverlodge operation; and Saskatchewan Research Council for the tour of their laboratories and pilot plant facility.

CANADIAN URANIUM PRODUCERS' METALLURGICAL COMMITTEE
1981 ANNUAL MEETING

MAY 19-20, 1981

SHERIDAN CAVALIER HOTEL, SASKATOON

PROGRAM

TUESDAY, MAY 19, 1981 *(Canadian Room)*

0900 *Development of Ranger Uranium in Australia*
- R. Ellerman (Wright Engineers)

0945 *Midwest Project - Canada Wide Mines*
- L. Melis (Canada Wide Mines)

1030 *Coffee*

1100 *Collins Bay B Zone Development*
- A. Moorish (Gulf Minerals)

1145 *Eluex Alternatives for Madawaska*
- G.M. Ritcey (CANMET

1230 *Lunch*

1400 *Preconcentration of Uranium Ores - Economic Assessment of CANMET Work*
- I.C. Edwards (Lummus)

1445 *Chloride Processing - Economics Assessment of CANMET Processes*
- I.C. Edwards (Lummus)

1530 *Coffee*

1600 *Canadian Participation in the NEA/IAEA Working Group on Uranium Extraction*
- M.C. Campbell (CANMET)

1930 *Dinner (Cocktails at 1900) - Maple Leaf Room*

WEDNESDAY, MAY 20, 1981 *(Canadian Room)*

0930 *Report on National Tailings Committee - V.A. Haw (CANMET)*

1015 *Coffee*

1045 *Saskatchewan Research Council - Pilot Plant and Facilities Description - W. Husband (SRC)*

1130 *Business Meeting*

1400 *Tour of SRC Analytical and Pilot Plant*

THURSDAY, MAY 21 and FRIDAY, MAY 22, 1981

Optional Tour of Gulf Rabbit Lake and Eldorado Nuclear Operations by Charter Aircraft. This tour is limited to 15.

CUPMC ATTENDANCE, MAY 19-20, 1981

Name	Affiliation
J.C. Anderson	Key Lake Mining Corporation
G. Angelov	G. Angelov & Associates
A.W. Ashbrook	Eldorado Nuclear Limited
B. Bihari	Rio Algom Limited
D.W. Bosch	A.H. Ross & Associates
M.C. Campbell	CANMET
A.B. Carino	Gulf Minerals
S. DiCarlo	Gulf Minerals
P. N. Dhir	Gulf Minerals
R. Ellerman	Wright Engineers
J.W. Fisher	Consultant
C. Freitag	Kilborn Engineering
K. Fraser	Canada Wide Mines Ltd. (Esso)
J. Goode	Kilborn Engineering
W.A. Gow	CANMET
V.A. Haw	CANMET
L. Heyman	Agnew Lake Mines
W. Husband	Saskatchewan Research Council
I.J. Itzkovitch	Eldorado Nuclear Limited
R. John	Eldorado Nuclear Limited
E. Krause	INCO
P. Kresin	Madawaska Mines Limited
V.I. Lakshmanan	Ontario Research Foundation
R. McClure	Eldorado Nuclear Limited
L.A. Melis	Canada Wide Mines Ltd. (Esso)
W. Milde	Gulf Minerals
A. Moorish	Gulf Minerals
M. Nevin	Key Lake Mining Corporation
R. Oberholtzer	Eldorado Nuclear Limited
A. Oliver	Eldorado Nuclear Limited
G.M. Ritcey	CANMET
B. Rosner	Key Lake Mining Corporation
K. Sarkar	Key Lake Mining Corporation
H. Sawatzky	CANMET
J. Schisler	Eldorado Nuclear Limited
W. Spyker	Saskatchewan Research Council
R. Weir	Sherritt Gordon
L. Wilson	Gulf Minerals
M.K. Witte	Ontario Research Foundation
K. Wyper	Gulf Minerals
W. Young	Saskatchewan Mining Development Corporation

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The Future of the Rabbit Lake Mill of Gulf Minerals Canada Limited
- A. Moorish .... 3-1 - 3-11
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Development of an Eluex Process for the Treatment of High Chloride Uranium Eluates - G.M. Ritcey .... 4-1 - 4-20

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THE RANGER URANIUM PROJECT

A GREEN FIELD DEVELOPMENT

by R. Ellerman

Wright Engineers Limited

June 1981

I. INTRODUCTION

The Ranger development story spanned a period of 11 years from the date the orebody was first discovered in 1970 to the commencement of full scale Production in September 1981. The early establishment of the size and grade of the ore reserve showed it to be a major discovery of significant economic importance to Australia. Yet, in the intervening years, external forces both government and pressure groups, caused frustrating delays to its development. In retrospect however, if one ignores economic factors, it did provide a period to adequately plan the development programme for the future. Ranger is the first of the major uranium projects proposed for Australia over the next decade. These developments will have a significant impact on the Australian economy, the world's uranium marketing scene and nuclear power development. In terms of cost of production, its' only rival will likely come from the Saskatchewan uranium province.

The ownership of Ranger originally rested with two well-known and large Australian mining companies, Peko-Wallsend and E.Z. industries. In 1974, the Australian Labour Government acquired 50% interest in Ranger by an agreement to put up 72.5% of the capital development cost. This relationship lasted until 1979 at which time the present Government sold off all its share interest to a new company formed by the original principals. This new publically owned company, holding 100% interest in Ranger, is Energy Resources of Australia Ltd. Major shareholders are Peko-Wallsend and E.Z. Industries, each owning 30.8% of the issued shares. Japanese and German interests hold minority positions (24.2%) while the remaining shares (14.2%) are held by the public.

Wright Engineers Limited first became involved in the Ranger Project in 1973 as consultants in revisions to the pre-feasibility study and updating of the capital and operating cost estimates. During the latter part of 1976 and early 1977, Wright Engineers Limited in association with Davy Pacific completed a final feasibility study in Australia covering all aspects of the project except mining. During these studies and subsequent developments, Ranger was, and continues to be served by A.H. Ross & Associates of Toronto as technical consultants.

With the completion of the Fox Commission Hearing covering the "Ranger Uranium Environmental Inquiry", in early 1977, the federal government gave Ranger the "green light" to proceed with the development of the project subject to the restrictive conditions as recommended in the Fox Reports. Ranger appointed Wright Engineers (Pty) Ltd. - Davy Pacific as their EPCM contractor to design and manage the construction of the project. Wrights had the responsibility for the design of the concentrator and ancillary services while Davy Pacific was responsible for project administration, purchasing and construction management. The project was be commissioned in August of 1981 and was essentially on schedule and within the capital budget estimate.

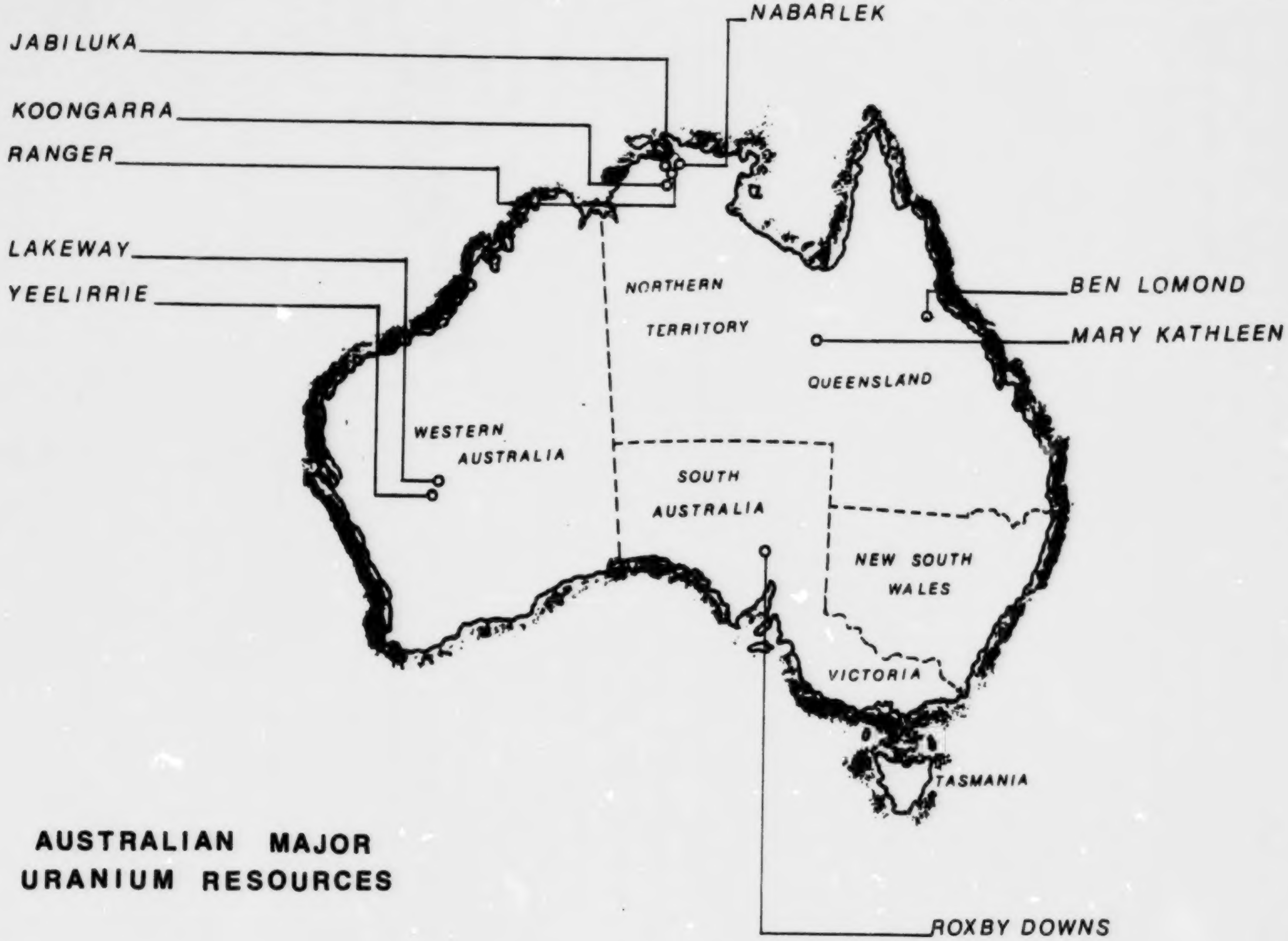
URANIUM IN AUSTRALIA

Australia produced only a relatively small quantity of U_3O_8 over the years since World War II, most of which came from the Mary Kathleen operation in northern Queensland. Known reserves up to 1970 were not of great significance. Discoveries in the Darwin area however, dramatically changed this situation. In that year and the following, four major strikes were made which quickly proved to contain in excess of 700 million lbs of U_3O_8 with ore grade in the range of 0.3%. At least one contained appreciable amounts of gold. Coincidental with this period, shallow calcrete deposits of uranium were discovered in Western Australia where Western Mining Corporation blocked out about 14 million lbs. of U_3O_8 having an ore grade of about 0.12%. This deposit, known as Yeelirrie, is now under active development. In South Australia the Roxby Downs deposit, owned jointly by Western Mining Corporation and B.P. Australia Ltd., will probably prove to be an exceptional reserve of U_3O_8. Ore reserves have not been published but one drill hole intersected the ore zone for 306 metres and contained 1.0% Cu and 0.35 lbs/tonne of U_3O_8. Estimations are that the operation could be developed to process 40,000 tonnes per day. A great many minor deposits have also been discovered throughout Australia but development of these have not been announced. Fig. 1 shows the location of the major deposits.

Table 1 summarizes the major uranium production potential in Australia for the 1980's. Whether this entire potential will be in production by the end of the decade is not known. Technically, there are no reasons why it cannot happen. Constraints of markets, politics and other external forces, however, will likely govern this schedule.

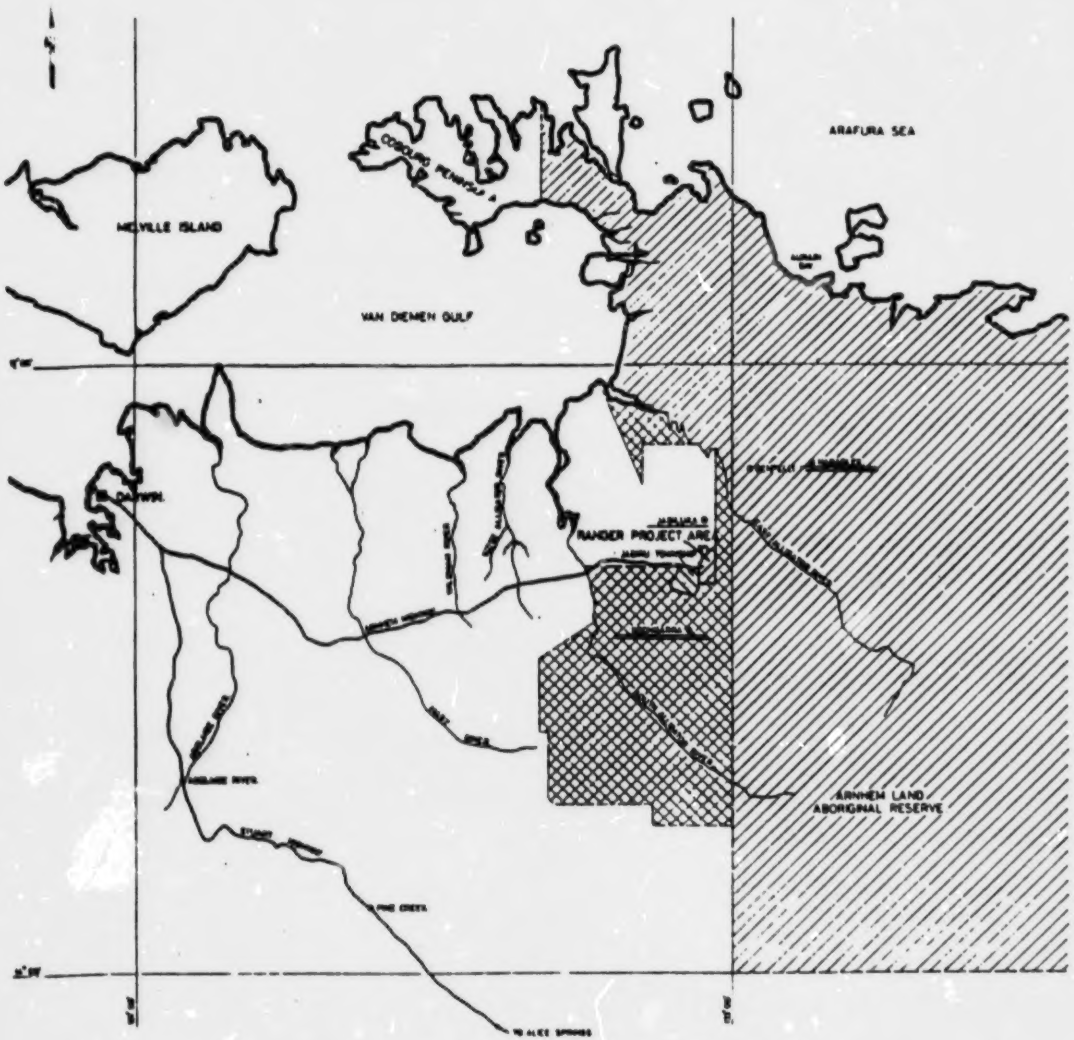
URANIUM IN THE NORTHERN TERRITORIES

The major share of Australian uranium reserves and production potential is presently located in the Northern Territories east of Darwin. The four major orebodies shown in Table II represent a 20-year potential reserve at a production rate of 37 million pounds per year. Nabarlek is now in successful production. Both Koongarra and Jabiluka are well into the final planning stage awaiting government approval to proceed. By the end of 1981, production from Ranger and Nabarlek will be at the rate of close to nine million pounds per year. Indications are that most of this production is sold under long-term contracts. Potentially, over 37 million pounds per year could be produced by mid to late 1980 from this area. Fig. II shows the location of these deposits.

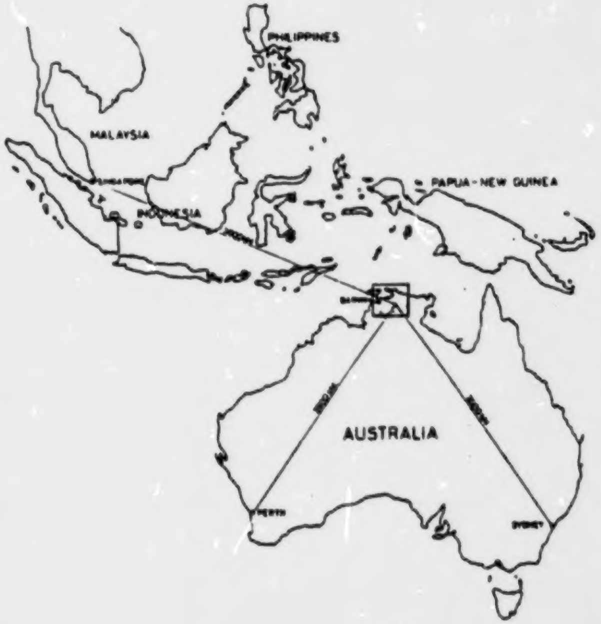


AUSTRALIAN MAJOR URANIUM RESOURCES

figure 1



ALLIGATOR RIVER REGION, NORTHERN TERRITORY



AUSTRALIA & SOUTH-EAST ASIA

FIGURE 2

PROJECT 1253
WRIGHT ENGINEERS - [illegible] PACIFIC

RANGER URANIUM MINE
LOCATION PLAN

1253-70-A0005

TABLE I

AUSTRALIAN URANIUM POTENTIAL FOR THE 1980's

DEPOSIT	COMPANY CONTROL	DATE DISCOVERED	RESERVES lbs U_3O_8	EST. GRADE % U_3O_8	ESTIMATED PLANNED PRODUCTION RATE - lbs U_3O_8/YR
Koongara	Denison Mines Ltd.	1970	33,000,000 Plus	0.3 to 0.35	2,200,000
Jabiluka	Pancontinental Mining Ltd.	1971	457,000,000	0.3 to 0.35	Initial 6,600,000 Final 20,000,000
Ranger	Energy Resources Australia Ltd.	1970	220,000,000	0.30	Initial 6,600,000 Final 13,200,000
Nabarlek	Queensland Mines Ltd.	1970	26,500,000	2.40	2,200,000
Yeelirrie	Western Mining Corp. Ltd.	1972	100,000,000	0.14	5,500,000
Mary Kathleen	Mary Kathleen Uranium Ltd.	1954	14,000,000	0.12	1,350,000
Total Potential			850,000,000	0.37	44,450,000
Roxby Downs	Western Mining Corp. Ltd.	1975-76	Huge	Est. 0.02 Plus Cu + Au	Unknown Est. 40,000 tpd Ore

TABLE II

NORTHERN TERRITORIES POTENTIAL PRODUCTION OF URANIUM

1980's

DEPOSIT	COMPANY CONTROL	DATE DISCOVERED	ESTIMATED RESERVES lbs $U_2 0_8$		GRADE % $_3 0_8$	PLANNED PRODUCTION lbs $U_3 0_8$/YEAR
Koongara	Denison Mines Ltd.	1970	33,000,000	Plus	0.30	2,200,000
Jabiluka	Pancontinental Mining Ltd.	1971	457,000,000		0.35	20,000,000
Ranger	Energy Resources Australia Ltd.	1970	220,000,000		0.30	13,200,000
Nabarlek	Queensland Mines Ltd.	1970	26,500,000		2.40	2,200,000
	TOTAL POTENTIAL		736,500,000		0.41	37,400,000

II. THE RANGER PROJECT

GEOLOGY

The geology of the Ranger Project area is complex and can best be summarized by quoting from the "Prospectus" of Energy Resources of Australia Ltd. This summary was prepared by Geopeko, a subsidiary of Peko Wallsend Ltd.

1. Introduction

The uranium deposits described in this report and in this Prospectus as the Ranger Deposits are located at longitude 132°55'E, latitude 12°42'S and are within the Ranger Project Area (See Fig. III). This area lies 265 km by sealed road east from Darwin. Road access may be cut for up to two weeks of the year during the monsoon season from December to March.

Interest in the region developed because the geological environment appeared similar in some respects to that of the Rum Jungle uranium deposits located 80 km south of Darwin.

Initial reconnaissance surveys were undertaken in 1968. An airborne spectrometer survey in October 1969 revealed an area of high gamma radiation which was located by a ground party in November 1969. This anomaly was later found to be part of a zone of distinct radiometric anomalies extending over a distance of approximately 6 km. These anomalies were numbered 1 to 5. Anomaly 2 is now outside the Ranger Project Area.

Drilling commenced in July 1970 on the most intense anomaly, designated No. 1 Anomaly (now No. 1 Orebody). Drilling of No. 3 Anomaly (now No. 3 Orebody), commenced in October 1971, and both programmes were suspended at the end of 1973. Ore blocking of No. 1 Orebody recommenced in August 1977 and a final reassessment of the reserves of this body was completed in June 1978. Ore blocking of No. 3 Orebody is still in progress.

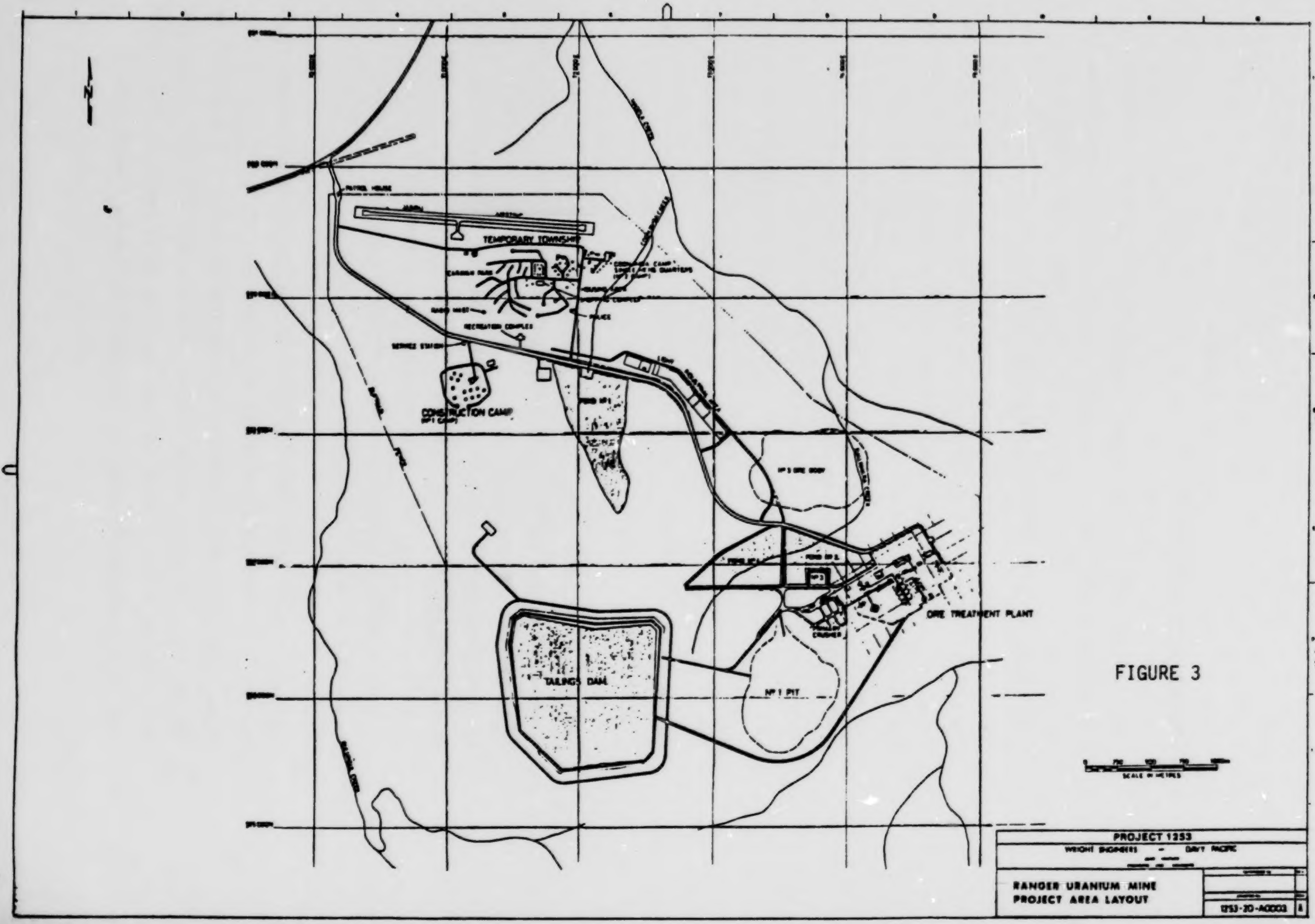

N
PATROL HOUSE
AIRSTRIP
TEMPORARY TOWNSHIP
CARAVAN PARK
RADIO MAST
RECREATION COMPLEX
SERVICE STATION
CONSTRUCTION CAMP
(NO 1 CAMP)
POLICE
POND Nº 1
POND Nº 2
Nº 3 ORE BODY
ORE TREATMENT PLANT
CRUSHER
Nº 1 PIT
TAILINGS DAM
SCALE IN METRES
PROJECT 1253
WRIGHT ENGINEERS - DAVY PACIFIC
RANGER URANIUM MINE
PROJECT AREA LAYOUT
1253-20-A0003

FIGURE 3

2. Geology of the Ranger Project Area

The local geology in the Ranger Project Area has been established in considerable detail. Some 1250 auger holes, 250 rotary/percussion holes, and more than 120 diamond holes have been drilled for exploration, ore delineation, and site investigation purposes since 1969.

The ore bearing rocks known locally as the Mine Sequence trend north-south. The lower part of the sequence is predominantly carbonate rocks (magnesite and dolomite) with minor interbedded schists called the Lower Mine Sequence. Overlaying this is the Upper Mine Sequence, which is predominantly a sericite-phlogopite-quartz-feldspar microgneiss when it is found away from the ore environment. Locally it contains graphite and other forms of carbon. Above the microgneiss, another irregularly developed dolomite horizon is present. The overlying rocks, the Hanging Wall Sequence, consist essentially of coarse grained serecite schists, and thin layers of amphibolite. This sequence persists to the eastern boundary of the Ranger Project Area.

The Ranger Project Area has been recently exposed by retreat of the sandstone escarpment, and it has been weathered to varying depths. Many parts are now covered with alluvial sand and silt, but some areas have residual soils which were derived directly from the bedrock. Where these bedrock-derived soils occur over ore deposits, such as at No. 1 and No. 3 orebodies, very large radiometric anomalies exist. However if the orebodies or bedrock-derived soils are covered by even a thin layer of sand or alluvium, most or all of the radiometric character is obscured.

Sand and alluvial soils of the Magela Creek system extend northwards from the No. 3 orebody for about 8 km to the boundary of the Ranger Project Area. This cover would prevent the detection of radioactive anomalies by surveys from the air or on the surface. Gravity and magnetic surveys and some limited probe drilling indicate that the Mine Sequence may extend continuously beneath the cover in this part of the Ranger Project Area.

The Mine Sequence appears to be displaced to some extent by faulting, but information about its position, structure and mineralization is very incomplete. Very detailed exploration to penetrate beneath the soils and Mesozoic sands will be necessary to determine its potential. This may be a long and costly programme, but it is strongly recommended in view of the inherent uranium content of the Mine Series rocks, and the possibility, of structures favourable to the concentration of uranium mineralization which might be found beneath the overburden.

3. Geology of the No.1 Orebody

No 1. Orebody occurs in rocks of both the Upper Mine Sequence and the Lower Mine Sequence. Significant changes have affected the rock types within the ore zone which is also surrounded by an alteration halo.

Two main types of alteration are important. The carbonates are silicified, and chlorite (a soft dark coloured mica) replaces most other minerals present. In particular, it replaces the feldspars and other minerals rich in iron and magnesium. Section 10,100N (Fig. V) is a typical east west cross-section through the deposit. This illustrates how the orebody is predominantly within the schists of the Upper Mine Sequence, and extends down into the Lower Mine Sequence. The Upper Mine Sequence, however, is altered to a sericitequartz-chlorite schists in the ore zone, grading to unaltered microgneiss to the east. The carbonates of the Lower Mine Sequence to the east have been completely replaced by fine grained silicious rock or chert, and masssive vein type chlorite rock. Within the ore zone a marker shcist horizon persists through the zone of alteration. All rocks within the ore zone are brecciated (broken and history of movement and change which is probably related to the mineralizing process).

There is a reduction of thickness of the Lower Mine Sequence beneath the orebody, and it is postulated that the localization of the mineralization has been caused by disturbance of the Lower Mine Sequence, allowing the formation of suitable structures for percolation and concentration of ore

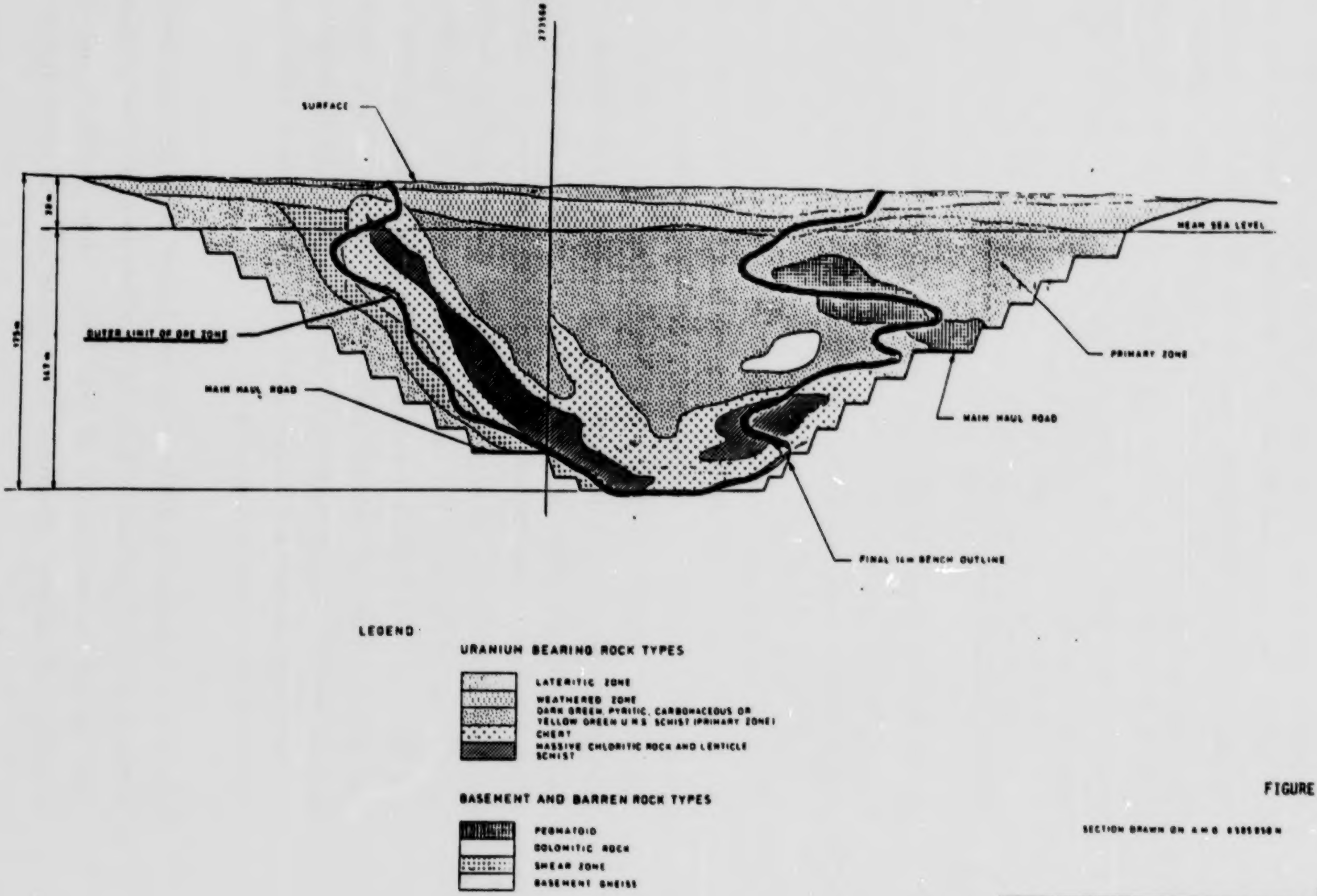


FIGURE V

SECTION DRAWN ON A M G 8585850 N

DRAWING No 5

1 1000

RANGER URANIUM MINES PTY. LTD
WRIGHT ENGINEERS – DAVY PACIFIC

SIMPLIFIED CROSS SECTION
THROUGH CENTRE OF Nº1 OREBODY

1253-11-A0003

TABLE III

RANGER ESTIMATED ORE RESERVES

OREBODY	TONNES	GRADE % U_3O_8	TONNES U_3O_8	STRIP RATIO
No. 1	15,000,000	0.325	48,750	3.4:1
No. 2	22,000,000	0.22	48,400	?
TOTAL	37,000,000	0.26	97,150	?

forming fluids within it and also in the overlying schists. The abundant chloritic veinlike structures within the Lower Mine Sequence carry considerable amounts of uranium mineralization. There appears to be a link between this chlorite and the pervasive uranium mineralization in the overlying schists.

The primary uranium mineralization is almost entirely pitchblende, but there are also traces of brannerite. The pitchblende occurs both as disseminations in chlorite veins and as coatings on fractures in about equal proportions. The sulphide content of the uranium orebody is low, approximately 0.6% weight.

Weathering has resulted in some redistribution of the uranium mineralization, and complex secondary uranium minerals such as saleeite, metatorbernite, and sklodowskite have been formed. Although there has been minor down slope migration in the soild profile, there is no evidence that uranium has been removed to any significant extent because of the oxidation of pitchblende. Indeed it appears that there are sufficient substances such as phosphates within the soil profile to fix the uranium soon after it is oxidised.

MINING

No. 1 Orebody, containing approximately 17,000,000 tonnes of recoverable ore, has a shape which makes it well suitable to open-cut mining methods. Essentially the ore zone has the dimensions of 400m x 200m x 175m deep. The above ore tonnage can be extracted with a waste to ore ratio of 3.4 : 1 by conventional open-cut methods. Estimated rcoverable reserves using a 0.1% U_3O_8 cut-ff grade is:

	Ore Tonnes	Grade	Tonnes U_3O_8
Lateritic	848,000	0.395	3,350
Weathered	1,456,000	0.399	5,809
Primary	13,125,000	0.332	43,575
Dilution	1,543,000	0.030	463
Total	16,972,000	0.313	53,197

The pit is designed using 7 m bench heights suitable for front-end loader operation. Pit slopes to the ultimate depth of 175 m range between 35 and 45 degrees. Dewatering pumps can handle up to 6,000 m^3/d during the wet season. This water is stored in No. 3 retention pond for use as mill process water or by-passed to the tailing pond.

Three ore stockpiles will be developed in close proximity to the primary crusher containing, high, medium and low-grade ores to assist in grade control into the mill or be available during interruptions to routine mining.

Ore from the upper lateritic zone, containing slime forming constituents, has now been stockpiles and will be consumed at a controlled rate during the life of the operation. This stockpile contains about 800,000 tonnes of ore having a grade of about 0.40% U_3O_8.

Ore leaving the pit in trucks is evaluated by an overhead scanning device to determine the U_3O_8 grade. Normally, the ore is dumped directly into the primary crusher but may be diverted to one of the three stockpile to maintain daily grade control.

ORE TREATMENT PLANT

The Phase I ore treatment plant has been designed to produce 3,000 tonnes (6,600,000 lbs) of U_3O_8 per year from the processing of 1.15 million tonnes of ore having an average U_3O_8 content of 0.3% (Table IV). This corresponds to a treatment rate of 3,500 tonnes per day or 146 tonnes per hour. Phase II will double the plant capacity.

The ore treatment process includes, primary, secondary and tertiary crushing, rod and ball mill grinding, acid-oxidization leaching, liquid-solid separation by CCD washing thickeners, pregnant solution clarification, solvent extraction and yellow-cake precipitation, drying and packing. The process flowsheet is shown in Fig. VI.

Figures VII & VIII show the general plant site area including the Ranger townsite location in relation to the project area. Table V outlines the general climatic conditions of the area. Fig. IX shows the general arrangement of the plant site area.

TABLE IV

RANGER URANIUM PROJECT - PRODUCTION CRITERIA

PHASE 1

U_3O_8 Production	-	3,000	t/y	(6,600,000	lbs/y)
		9,000	kg/d	(20,000	lbs/d)
Head Grade	-	0.3% U_3O_8			
Recovery of U_3O_8	-	88.3% Plus			
Mill Rate - Ore	-	1,150,000	t/y		
	-	3,500	t/d		
	-	146	t/h		

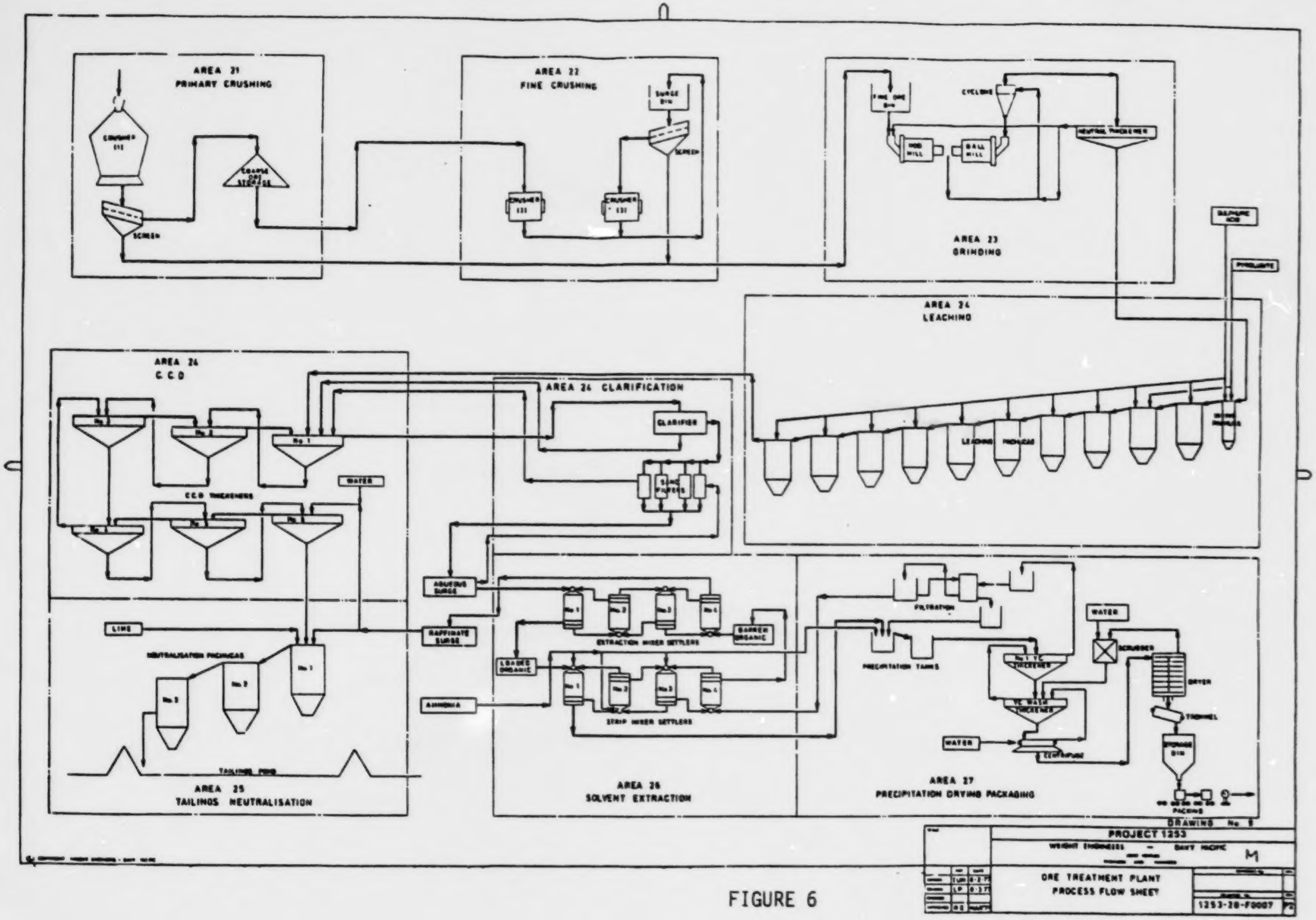

AREA 21
PRIMARY CRUSHING
CRUSHER 111
COARSE ORE STORAGE
SCREEN
AREA 22
FINE CRUSHING
SURGE BIN
SCREEN
CRUSHER 131
CRUSHER 131
AREA 23
GRINDING
FINE ORE BIN
ROD MILL
BALL MILL
CYCLONE
NEUTRAL THICKENER
SULPHURIC ACID
PYROLUSITE
AREA 24
LEACHING
LEACHING PACHUCAS
AREA 24
C C D
CCD THICKENERS
No 1
No 2
WATER
AREA 24 CLARIFICATION
CLARIFIER
SAND FILTERS
AQUEOUS SURGE
RAFFINATE SURGE
AMMONIA
LIME
NEUTRALISATION PACHUCAS
No 1
No 2
No 3
TAILINGS POND
AREA 25
TAILINGS NEUTRALISATION
EXTRACTION MIXER SETTLERS
STRIP MIXER SETTLERS
BARREN ORGANIC
LOADED ORGANIC
No 1
No 2
No 3
No 4
AREA 26
SOLVENT EXTRACTION
FILTRATION
PRECIPITATION TANKS
No 1 YC THICKENER
YC WASH THICKENER
WATER
CENTRIFUGE
WATER
SCRUBBER
DRYER
TROMMEL
STORAGE BIN
PACKING
AREA 27
PRECIPITATION DRYING PACKAGING
DRAWING No 8
PROJECT 1253
WRIGHT ENGINEERS — DAVY PACIFIC
M
ORE TREATMENT PLANT
PROCESS FLOW SHEET
1253-28-F0007

FIGURE 6

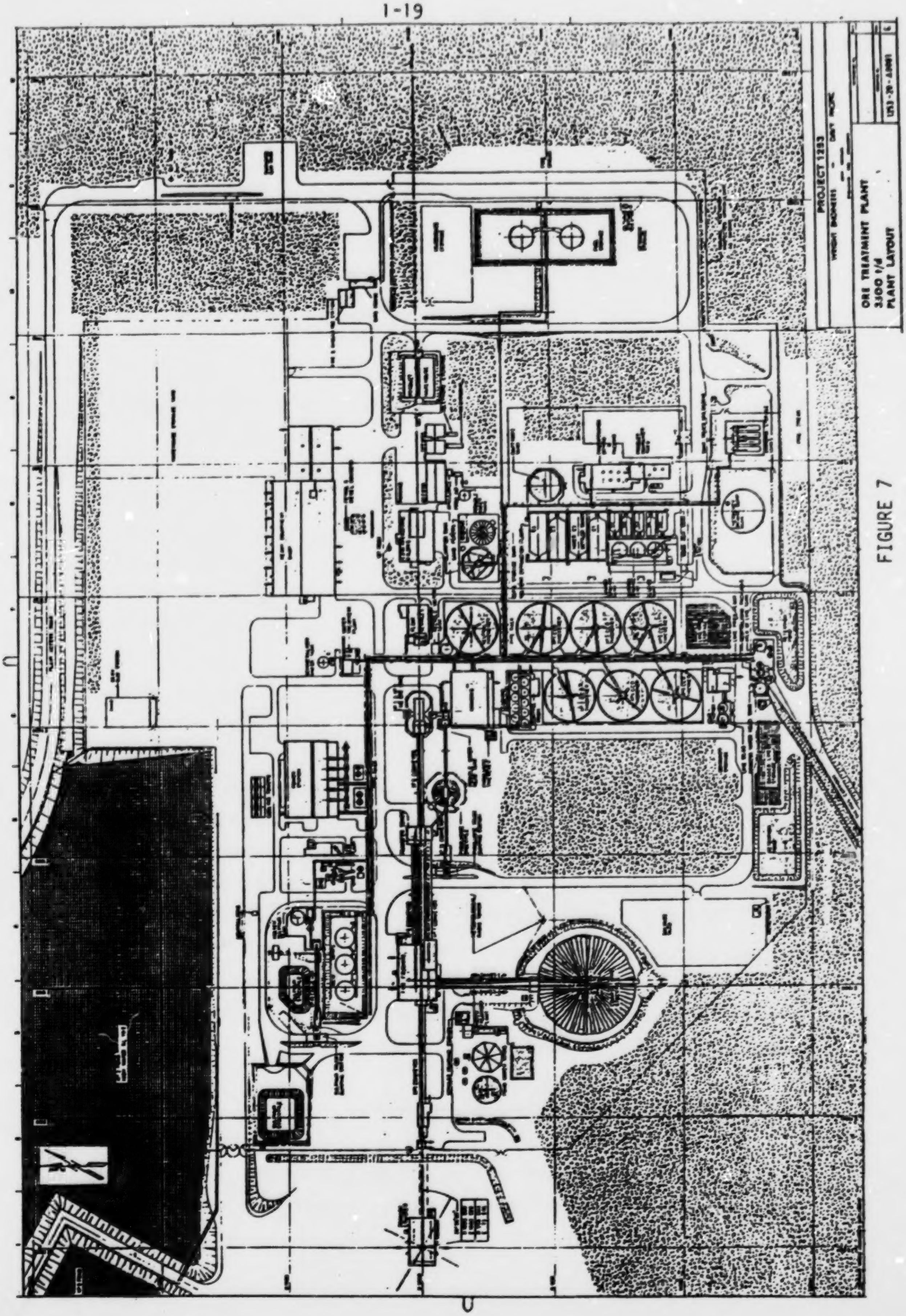


FIGURE 7

1. Primary Crushing

The primary crusher was sized to accommodate the large rock pieces asociated with open-cut mining. For this reason, a 54" x 74" gyratory unit was selected. While the crushing capacity of this unit, at a setting of 8 inches, is in excess of 2,000 t/h, we anticipate the nominal put-through will not exceed 800 t/h based on 6 hours per day of operation.

The gyratory crusher unit is mounted in a concrete structure designed to utilize gravity flow through the system. Ore delivered by truck from the open-cut mine or from the stockpiles is dumped directly into the crusher chamber where it is reduced in size to about minus 400 mm (8") and then falls into a receiving bin. From the bin the ore is withdrawn at a controlled rate by a 1500 mm (60") apron feeder to a 1200 mm (48") belt conveyor for transportation to an 8' x 20' double-deck vibrating screen. About 15% of the ore flow is removed as a finished minus 19 mm product and is conveyed to the fine ore bins. The plus 19 mm product from both screen decks is collected and transported by a 1200 mm (48") conveyor to an open, conical, coarse ore stockpile.

Operation of the primary crusher follows the mining schedule of five days per week, one shift per day, eight hours per shift. One operator controls the circuit from a central control room overlooking the crusher chamber. The control panel incorporates all the necessary elements for operation of the circuit including pushbutton layout, instruments and alarms, mimic flow diagram display, TV monitors and the programmable logic controller (PLC) display unit. The PLC system is incorporated into the instrumentation using programmed sequential logic control for starting and shutdown of all major equipment within the primary crusher circuit.

2. Secondary and Tertiary Crushing and Screening

Coarse ore from the primary crusher stockpile is crushed to minus 19 mm (3/4") utilizing an 2100 mm standard crusher in open circuit to an 2100 mm short head crusher in closed circuit with two 2400 x 6000 mm (8' x 20') single deck vibrating screens. Screen undersize is conveyed to two 5,500 live tonne fine ore bins.

For Phase I operations, the plant is scheduled to operate one shift per day, seven days per week at a daily tonnage rate of 3,000 tonnes (500 t/d is derived from the primary crushing circuit). The plant is enclosed in a building which also houses the primary screen.

The ore is reclaimed from underneath the coarse ore stockpile with one or more of three vibrating pan feeders drawing from chutes located in the roof of the reclaim tunnel. The feeder size is 1200 mm x 2100 mm (48" x 96") and they are located in-line, all discharging onto Conveyor No. 3. The ore is transported into the fine crushing building with Conveyor No.3 which transfers onto Conveyor No. 4, a short retractable conveyor which feeds the secondary crusher.

The ore discharges from Conveyor No.4 directly into the feed chamber of the secondary crusher. This crusher is an 2100 mm (84") cone crusher with a closed side setting of 32 mm. The product from the secondary crusher is discharged onto Conveyor No.5 and is transported via Conveyors No.6 and 7 to a surge bin located above the secondary screens. The ore from the surge bin is withdrawn by two 1500 mm x 2100 mm (60" x 84") vibrating pan feeders. Each feeder discharges onto an 2400 mm x 6000 mm (8' x 20') single-deck screen with 19 mm x 38 mm openings. The undersize from the two screens falls onto Conveyor No. 9 where it joints the undersize from the primary screen and is transported to one of the two fine ore storage bins. The oversize from both secondary screens falls onto a short horizontal retracable Conveyor No.8, which feeds the Tertiary Crusher. The Tertiary Crusher is an 2100 mm (84") cone crusher with a closed side setting of 16 mm. The tertiary crusher product joins the secondary product and is recycled back to the secondary screens in a closed circuit.

Two fine ore storage bins are provided to store the 19 mm crushed material each with a capacity of 5,500 live tonnes. Conveyor No.9 transports the primary and secondary screen undersize to the fine ore bins and discharges into a 2-way chute. The chute is provided with a 2-position gate which allows Fine Ore Bin No.2 to be filled directly when the gate is in position No.1. In position No.2 the gate allows the fine ore to discharge onto a short horizontal conveyor (No.10) which feeds directly into Fine Ore Bin No.2.

Operation of the fine crushing and screening plant is normally carried out by one operator working from the elevated control room located at the screening floor level. An operator control panel is installed in the fine crushing plant control room where all the necessary information is displayed for the normal and safe operation of the plant equipment. The control panel comprises:

a) Pushbutton Layout
b) Indicators, Indicating Controllers and Recorders
c) The 'Total Distributed Control System 200' (TDC-2000) Data Entry Panel
d) Mimic Diagram Display
e) The 'Programmable Logic Controller' (PLC) Visual Display Unit

3. Grinding

Grinding of the ore to produce a product of 80% passing 174 microns is accomplished with a rod mill in open circuit with a ball mill in close circuit with cyclones. Cyclone overflow passes to a thickener whose underflow becomes feed to the leaching circuit. Operation of the grinding plant is continuous, seven days per week. The grinding circuit is enclosed in a building which also contains the pyrolusite grinding circuit and flocculant make-up.

Fine ore (-19 mm) is withdrawn at a controlled rate from each bin through two slot feeders located in the bin bottom. The ore discharging from each slot is carried by a variable speed conveyor to a common conveyor feeding the rod mill. Control of the feed rate is by weightometer which also ratios the flow of water to the rod mill. Fine ore is discharged direclty into an 3300 mm x 4800 mm (11' x 16') rod mill equipped with a 705 kW motor (950 hp) and trommel discharge. Rod mill discharge flows by gravity to a cyclone feed pump box which is common to both the rod and ball mill cyclone underflow flows by gravity to the feed opening of a 4100 mm Ø x 4900 mm ball mill equipped with a 1300 kw motor Ball mill discharge flows to the cyclone feed pump box by gravity. The grinding control system is based on mass flow to three 500 mm (20") cyclones utilizing density and flow measurements combined with variable speed pumping. Percent solids of the feed to the cyclones is kept constant by water addition to the pump box controlled by the density guage. A level sensor in the pump box controls the speed of the pumps. The combined flow and density measurements are computed into mass flow which in turn controls the rate at which ore is fed to the rod mill. The mass flow parameter can be pre-set to the desired value for the operation.

Grinding plant parameters are:

Ore Treatment Rate:	1,1500,000	t/y
	3,500	t/d
	146	t/h
Availability:	90%	
Feed K_{80}:	19	mm
Product K_{80}:	174	microns
Rod Mill WI	17	kWh/t
Ball Mill WI	18	kWh/t

Cyclone overflow containing about 20% solids flow by gravity to the feedwell of a 39 m diameter neutral thickener located adjacent to the grinding plant. To aid settling, a flocculant is added at the rate of about 0.02 - 0.04 kg/t ore. Thickener overflow flows by gravity to a holding tank from where it is pumped to the grainding circuit as make-up water. Thickener underflow containing about 60% solids is withdrawn by diaphragm pump and then flows by gravity to the leach feed pump box. From here, the slurry is pumped to the head of the leaching circuit.

Operation of the grinding area including the pyrolusite handling and processing system is normally carried out by one operator working from a control room located at the mill operating floor level in the grinding building. An operator control panel is installed in the control room where all the necessary information is displayed for the normal and safe operation of the plant equipment. The control panel comprises:

a) Push Button Layout
b) Mimic Diagram Displays
c) The "Programmable Logic controller" (PLC) Visual Display Unit
d) TDC-2000 Operator Station and VDU, for Control of Grinding Circuit
e) TDC-2000 Operator Station and VDU, for Total Plant.

4. Leaching, CCD Thickening & Tailing Neutralization

Ore slurry from the neutral thickener, containing about 60% solids, is pumped to the mixing pachuca tank. Here, under vigorous agitation, the bulk of the sulphric acid required for leaching is added. From the mixing pachuca the slurry flows by gravity to a single series of 10 leach pachuca tanks, 7 m diameter by 12 m vertical side wall plus a 60° cone, to give a total retention time of about 30 hours. Additional acid is added under automatic control during leaching to maintain an average pH of about 1.8 to 2. Total acid consumption is expected to be 40 to 50 kg/t of ore. To maintain a redox potential of the pulp between -470 to -480 m V, pyrolusite is added under automatic control to any 2 of the first 5 tanks as a slurry. Pyrolusite consumption is expected to be between 7 to 11 kg/t. During normal leaching, the pulp temperature is expected to rise to about 70°C, the optimum operating temperature. To maintain this temperature, provisions are also made to add steam by direct injection under automatic temperature control.

Pulp from the final leach pachuca flows by gravity to a drop tank that absorbs the energy prior to being distributed to either No. 1 or No. 2 CCD washing thickener. Overall leach extraction of U_3O_8 is in the range of 92 to 94 percent.

The CCD washing circuit consists of six, 39 m diameter centre-pier type thickeners laid out in two parallel rows with a 7 m centre aisle. The thickeners are supported on steel structures above grade to eliminate tunnels and underground pipe channels. The centre ailse contains all necessary pumps, pump boxes and interconnecting piping and 4 m raised truck access way.

The CCD circuit is a typical countercurrent system that produces about 0.5% soluable U_3O_8 loss to tails. The settling area is about 0.34 m^2/t/d. Feed to each thickener is maintained at 15% solids while the underflow is withdrawn at about 55% solids by diaphragm pumps. Thickener overflow is advanced by gravity to the pump box receiving slurry from the preceeding thickener and both are pumped to their respective thickener by horizontal centrifugal pumps. To maintain 15% solids in the thickener feed, recycle overflow is added to the pump box. Flocculant addition to the thickener feed is at the total rate of 0.03 kg/t of ore, of which 0.02 kg/t is added to the feed to No. 1 thickener. Wash solution introduced into No. 6 thickener feed is solvent extraction raffinate and water as required.

No. 1 thickener overflow becomes the unclarified pregnant solution containing about 0.8 g/l U_3O_8 at a flow rate of 480 m^3/h. This solution contains up to 200 ppm solids.

Unclarified pregnant solution is pumped from No. 1 thickener to a conditioning box where recycle clarified solids, acid as required to control the pH at 1.8 and a flocculant are mixed prior to flowing by gravity to the feed-well of a reactor clarifier, 24 m in diameter x 9 m side-wall. Clarifier underflow solids are bled from the recycle stream to No. 1 CCD thickener. Clarified solution containing about 20 ppm solids is pumped to 4 pressure sand filters and thence to the clarified pregnant solution holding tank.

No. 6 CCD thickener underflow becomes the final tailings, and is pumped to the neutralization plant. This consists of 3 pachuca tanks in series each 5 m diameter by 12 m high plus a 60° cone. Residence time during neutralization is 2.5 hours. Milk of lime is added to No. 1 pachuca under pH control to maintain a pH of 8 in the discharge from No. 3 tank. This discharge becomes the final neutralized tailing and is pumped to the tailings pond.

Operation of the leaching, CCD, clarification and tailings areas is normally carried out by two operators working from a control room located at the CCD operating floor level next to the leach pachucas. An operator control panel is installed in the control room where all the necessary information is displayed for the normal and safe operation of the plant equipment. The control panel comprises:

a) Push Button Layout
b) Mimic Diagram Displays
c) TDC-2000 Operator Station and VDU, for control of leaching, CCD, clarification, lime preparation and tailings neutralization circuits.
d) Back-up Analogue Control Station.

5. Solvent Extraction, Precipitation and Drying

Purification and up-grading uranium in solution is accomplished by solvent extraction using a tertiary amine as the extractant. The resultant purified and concentrated uranium bearing solution is reacted with ammonia whereby the U_3O_8 is precipitated, dewatered and calcined to produce a product for shipment.

Clarified pregnant solution from storage is pumped at a controlled rate to the first of 4 extraction mixer-settlers where it is mixed with a countercurrent flow of organic solution consisting of 2.5% tertiary amine and 2.5% isodecanol in kerosine. Flows to the extraction mixer units are:

aqueous advance	-	467 m^3/h
organic advance	-	206 m^3/h
organic recycle	-	514 m^3/h
Total		1,187 m^3/h

O:A ratio mixer = 1.54 : 1
Organic loading = 2 g/l
Retention time- =
mixer = 1 min.

The effective extraction settler area is approximately 340 m^2 giving a total specific flow of 3.50 $m^3/h/m^2$ or an aqueous specific flow of 1.37 $m^3/h/m^2$.

Raffinate from the final extraction settler flows by gravity to a holding tank from where it is pumped to No. 6 CCD thickener as wash water.

Loaded organic solution from the first extraction settler flows by gravity to a storage tank from where it is pumped at a controlled rate to the first of 4 stripping mixer-settlers. The stripping solution in countercurrent flow to the organic contains 170 g/l of ammonium sulphate. Ammonia gas, injected under pH control, is used for pH adjustment during stripping. Flows to the stripping mixer units are:

aqueous advance	=	19 m^3/h
aqueous recycle	=	50 m^3/h
organic advance	=	206 m^3/h
Total		275 m^3/h

O.A. ratio mixers = 3:1

Aqueous loading = 27 g/l U_3O_8

The effective stripping settler area is 81 m^2 giving a total specific flow of 3.4 $m^3/l/m^2$.

Stripped organic from the final stripping settler flows by gravity to a storage tank from where is is pumped at a controlled rate to the last of the extraction mixers in a total recycle system. The loaded strip solution is pumped from the first stripping settler to the precipitation circuit.

Organic regeneration is performed in a single mixer-settler unit similar in size to the stripping units. Sodium carbonate solution used for regeneration. Each settler unit is equipped to allow the flooding-off of organic sludge. The sludge flows by gravity to a holding tank and treated at intervals for organic recovery which includes pressure filtration.

To minimize and/or prevent damage to the solvent extraction plant by fire, an elaborate protection system has been installed based on close liaison with fire insurance underwriters. The entire plant has been divided into five zones with concrete fire walls and 1 m perimeter walls separating the extraction circuit into two zones, the stripping circuit into two zones and the organic holding tanks into one zone. Each zone below the settlers and around the tanks is equipped with automatic foam stations. Each settler is equipped with a automatic organic dump system that can remove all the organic from each settler through a fire trap into a holding tank located outside of the solvent plant area. All tanks and settler covers are equipped with overhead water sprinklers that can be manually or automatically tripped in case of fire. Finally, a network of lightening arrestors are strategically located around the plant area.

The mixer settlers units, storage tanks and interconnecting piping, within the solvent extraction area is constructed of 316 L stainless steel. All storage tanks have SS covers while the settler tanks are covered with specially treated steel cladding supported on epoxy coated steel trusses.

The yellowcake precipitation circuit consists of two precipitation tanks in series followed by two stages of wash thickening. Loaded strip solution from the first stripping settler is pumped to the first precipitation tank where ammonia gas is injected under control to raise the solution pH to 7.2-7.6. Uranium precipitates and the slurry passes on by gravity to the second tank thence to the product thickener. Overflow from the product thickener is clarified and recycled to solvent extraction. Thickener underflow is withdrawn and pumped to the yellowcake wash thickener. Underflow from the wash thickener is pumped to a water washing centrifuge. The centrifuge cake is screw conveyed onto the first hearth of a 3250 mm diameter by 6 hearths multi-hearth dryer-calciner. The calcined product discharges into a trommel disintegrator and then by gravity to a 40 tonne storage - loadout bin. From this bin, the product is drummed for shipment.

The dryer-calciner exhaust gas is treated in a high energy venturi scrubber while the product storage and packing equipment is vented to a dry baghouse collector.

All tankage including thickeners, process piping and pumps are made of 316 L stainless steel. The centrifuge and venture scrubber is also of the same material.

Operation of all solvent extraction precipitation and drying plant is carried out by a single operator.

The operator control room is situated in the precipitation building. The operator control panel located in this room and all necessary information is displayed on the panel for normal safe operation of plant and equipment . The control panel consists of four main sections.

a) Push Button Layout
b) Mimic Diagram Displays

c) TDC-2000 console containing VDU and keyboard
d) Back-up Analogue Control Station.

6. Reagents

Sulphuric Acid - manufactured on-site at the rate of up to 190 t/d of 93% acid using a double catalysis plant burning elemental sulphur. Storage of up to 2400 t of sulphur is provided on-site while acid storage available is for 8500 tonnes in 3-15 m diameter x 10 m tanks.

Ammonia - storage of up to 450 tonnes is provided in 4 tanks. The system is complete with evaporator and compressor.

Lime - burned lime storage of up to 1600 tonnes is provided in two silos. Milk of lime preparation uses a 1500 mm x 2100 mm ball mill and necessary feeders and classifier. Milk of lime is stored as a 20% solids slurry in a 10 m diameter x 10 m agitated tank.

Pyrolusite - storage of up to 3500 tonnes is available. Pyrolusite is ground in a 1800 mm x 3000 mm ball mill in closed circuit with a cyclone. Cyclone overflow is thickened to 55% solids and stored in an agitated tank from where it is distributed in a loop line to the leach pachucas.

7. Dust and Fume Control

In the interest of the health and safety of workers and to meet the rigid environmental requirements as recommended in the Fox Commission Reports, special precautions have been taken for the control of airborne dust, radon gas and chemical fumes throughout the plant. As an example, in the grinding plant, all equipment that exposes the ground pulp to the atmosphere within the building are exhausted to stack to prevent radon contamination of the air. These include pump boxs, launders, mill discharge trommels, sumps etc.

Extensive dust and radon control systems have been incorporated into the crushing, conveying and stockpile circuits. The acid plant uses the double catalysis system, to minimize SO_2-SO_3 contamination of the environment. A great deal of design effort and cost has gone into making the plant a safe working environment limited only by the state of the art and technology available today.

8. Tailings Disposal

The tailings dam is located to the west of the plantsite and covers an area of 100 ha. It is sized to readily retain all the expected tailings from treading the ore from No. 1 ore body. The solids will be continuously flooded with about 2 metres of water to prevent radon emissions. Provisions have been made to reclaim water from the pond back to process.

UTILITIES

1. Power

Electrical power is generated on-site having a total installed capacity of 23 mw produced by 4 diesel generators of 5.2mw rating and 1 turbo-alternator of 2.5mw rating. Of the 4 diesel units, one will be on standby and a maximum of 3 operating. The diesel units are equipped with heat recovery whereby the exhaust gases pass through boilers to produce about 7000 kg/h low pressure process steam under normal load. In addition, the diesel cooling jackets produce hot water by exchanger and used for make-up in the total steam system. The turbo-alternator receives its high pressure steam supply from the acid plant and can be operated under a fully condensing or by-pass mode depending on power and steam demand.

2. Steam

Total steam consumption is estimated to vary between 7000 and 9000 kg/h and derived from the heat recovery systems and a standby package boiler at a pressure of 420 kPag (60 psi). Of this amount, up to 5000 kg/h will be used in leaching while other uses will be for sulphur melting, ammonia vaporization, boiler deaerators, etc.

3. Compressed Air

Three compressed air streams are produced on site. Low pressure air at 450 kPag (66 psi) is produced by 4 compressors, one of which is on standby, for a normal operating load of about 130 m^3/min (4500 CFM) for pachurce tank agitation. High pressure air at 750 kPag (110 psi) is produced by 2 compressors, one of which is on standby, for a normal load of 30 m^3/min (1000 CFM) for general distribution throughout the plant. Instrument air is produced at the rate of about 5.2 m^3/min (180 CFM) at a pressure of 750 kPag.

4. Water

Process and potable water is derived from two main sources. First, potable water comes from wells located about 15 km from the plant site near the South Aligator River. The capacity of this well system is sufficient to meet process water requires if and when required. Process water is mainly derived from run-off during the rainy season which is stored in a large reservoir for use during the entire year. Part of the process water requirements is derived by recycling raffinate as wash to the CCD thickener circuit.

CAPITAL COST

Published figures place the total capital cost of the Ranger project at $345 million. Of this amount about $65 million will have been spent on the township housing facilities. The project was completed within the budget estimate and on schedule.

ACKNOWLEDGEMENT

Wright Engineers Limited wish to express their thanks to Mr. Don Woods, General Manager, Ranger Uranium Project for permission to present the above information.

TABLE V

RANGER - CLIMATIC DATA

RAINFALL	4 Wet Months -	Dec. thru Mar.	-	1,160	mm
	4 Dry Months -	Jun. thru Sep.	-	10	mm
	Yearly Average		-	1,400	mm
TEMPERATURE	High Average -	October - April	-	38°C	
	Low Average -	July	-	16°C	
	Maximum High Recorded		-	41°C	
WIND SPEED	Maximum Recorded		-	145	km/h
	Maximum Expected		-	175	km/h
HUMIDITY	Average Maximum		-	95%	
	Average Minimum		-	39%	
	Lowest Recorded		-	16%	
EVAPORATION	Annual Average		-	2,000	mm
SUNSHINE	Average 4 yr Annual		-	8.2	h/d
	May thru Aug.		-	9.9	h/d
	Jan. thur Mar.		-	5.5	h/d

DISCUSSION OF PAPER BY R. ELLERMAN

L. Melis	What is the soluble loss?
R. Ellerman	0.5%
L. Melis	Why is the recovery low?
R. Ellerman	Probably due to some brannerite
L. Melis	Did you give any thought as to a central control room?
R. Ellerman	Yes
L. Melis	Any thought to use of automatic valves and gamma gauges?
R. Ellerman	Yes - Dorrco diaphragms were recommended by A. Ross but no automatic valves.
L. Melis	Do you foresee any difficulties because of climate on the maintenance?
R. Ellerman	No, adequate controls will look after the maintenance control.
M.K. Witte	What flocculant was chosen for the CCD circuit and does the lateritic ore give problems in autogenous grinding?
R. Ellerman	There are three different flocculents: 1 in neutral clarifier; 1 in CCD; 1 in ? Not certain of flocculents chosen. Regarding the lateritic ores, they do give problems in the grinding circuit and must be blended.
P.N. Dhir	Any work done with $NaClO_3$ and any difference between pyroluscite?
R. Ellerman	Warman investigated and checked pyroluscite, Caro's acid and chlorate. Pyroluscite was chosen on basis of economics and local availability.
L. Miles	What type of U scanner was chosen?
R. Ellerman	Amdel probe.

A. Oliver	How do you propose to apply the NH_3 to the stripping stages?
R. Ellerman	The ammonia will be injected by sparging into numbers 1 and 3, with pH control external to the mixer.
K. Fraser	What is the allowable limits on grade control and what has been done to control?
R. Ellerman	The limits are 0.2 - 0.4% for the design. The metallurgy showed no appreciable difference due to the grade handled.
J. Fisher	Did you take into account any control requirements for the environment guidelines?
R. Ellerman	The original flowsheet was designed prior to the Fox Commission without too many environmental concerns. Tailings pond will be flooded at all times. There is sufficient containment for 3-5 years.
E. Kraus	What are the problem elements? How much sulphides?
R. Ellerman	There are $<$0.5% S. Also have some soluble Si. Acid consumers are Ca and Al carbonates.
V. Haw	Is there any involvement in the laterites?
R. Ellerman	Upper zone 0.4% U_3O_8, compared to primary ore at 0.2-0.3% U_3O_8.
I. Itzkovitch	What is the regeneration stage for?
R. Ellerman	For picking up humic acids.
R. John	How many people are expected on the site?
R. Ellerman	350 approximately, including the mine.
I. Itzkovitch	Any problem with organic attacking the rubber.
R. Ellerman	Neoprene used, very little attack.
V. Lakshmanan	You mentioned humic acids and soluble silicates. Do you foresee any need for pre-treatment prior to the SX circuit?
R. Ellerman	We don't anticipate any problem. Humic acids are not a real problem. Silica is not known yet.

L. Melis	What is the current U marketing plan?
R. Ellerman	Production is 90% sold on long term contracts to Germany, Japan and Sweden.
J. Fisher	Any NH_3 removed?
R. Ellerman	No. There is a bleed to tails.
G.M. Ritcey	Who did the design?
R. Ellerman	Wright Engineering did, after the design of Bagdad Copper, with stainless steel tanks and piping. The solvent can be pumped in 12 minutes automatically to a holding tank. The plant incorporates a flood-off system for crud.
L. Heyman	Were there any considerations given to belt filters?
R. Ellerman	Test work indicated they would be too slow.

MIDWEST PROJECT

CANADA WIDE MINES LTD.

PRESENTATION FOR THE CANADIAN URANIUM PRODUCERS'
METALLURGICAL COMMITTEE MEETING

MAY 19 AND 20, 1981 SASKATOON, SASK.

L.A. MELIS, P. ENG.
SUPERINTENDENT-MILL DEVELOPMENT

K. FRASER
PROJECT METALLURGIST

J.W. FISHER, P. ENG.
CONSULTANT

MIDWEST PROJECT

CANADA WIDE MINES LTD.

1. INTRODUCTION

1.1 General

Canada Wide Mines Ltd., a subsidiary of Esso Resources Canada Limited, is planning the development of the Midwest uranium deposit in Northern Saskatchewan. Other participants in the project are Numac Oil and Gas Limited, Bow Valley Industries Limited, Mink Mining Corporation and Midwest Mining Corporation. The project office is based in Saskatoon.

Under the current schedule the mill is slated to start up in late 1986.

1.2 Location and Access

The Midwest uranium deposit is located in northern Saskatchewan, 370 km north of La Ronge and 700 km north of Saskatoon, (see Figure 1.1).

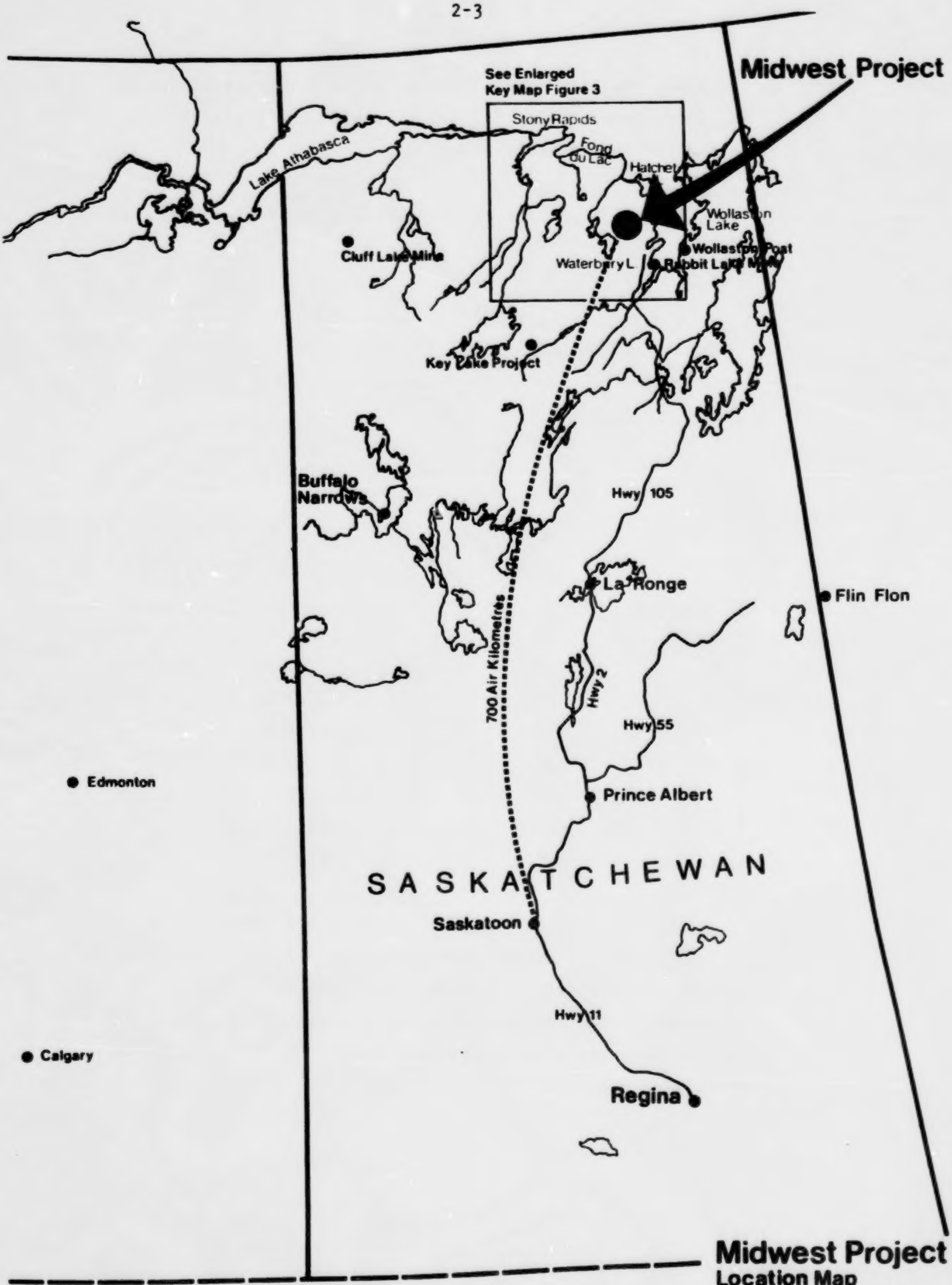
Access to the site is by plane landing on water or ice, or by winter road (27 km) from Highway #105, the highway from La Ronge to Rabbit Lake.

Construction of a permanent access road to the site is in progress with completion scheduled for July 1982. It will connect the Midwest site to Highway #105.

1.3 Deposit

The deposit occurs at a depth of about 200 m at the unconformity between nearly flat-lying sandstone and older underlying gneisses.

It consists of two ore types, massive high grade ore and a lower grade sandstone ore. The principal minerals in the massive ore are pitchblende (UO_2-UO_3), coffinite ($U(SiO_4)_{1-x}(OH)_{4x}$), gersdorffite (NiAsS), niccolite (NiAs)



Midwest Project
Location Map
Kilborn Figure 1.1

and rammelsbergite ($NiAs_2$), while the lower grade of ore is a friable sandstone containing clay with sooty pitchblende as the uranium mineralization.

Typical analysis of the expected mill feed is given in Table 1.1.

Table 1.1

Typical Mill Feed Analysis

U	-	1.9%	SiO_2	-	51.3%
As	-	3.3%As	Al_2O_3	-	10.0%
Ni	-	2.0%Ni	CaO	-	0.21%
Co	-	0.15%	Na_2O	-	.03%
Cu	-	0.13%	K_2O	-	2.25%
Pb	-	0.35%	Fe	-	2.25%
Zn	-	0.12%	MgO	-	1.66%
Bi	-	0.04%	Mn	-	.034%
Hg	-	<1 ppm	P	-	.055%
Mo	-	.058%	Ti	-	.55%
Se	-	18 ppm	Cr	-	.022%
S	-	1.91%	V	-	.13%
Pb 210	-	5620 pCi/g	Ra 226	-	8662 pCi/g
Pb 210	-	5500 pCi/g	Th 230	-	7505 pCi/g

In addition to the ore, a halo of low grade sub-economic material (special waste) lies above the deposit. This special waste totals approximately 2 million tonnes assaying 0.04%U. As this material contains some arsenic it has the potential for impact on the environment and will be stockpiled separately.

2. PROCESS

2.1 General

The flowsheet that is proposed at this time is the result of testwork that has been carried out over the last two years at Lakefield, Colorado School of Mines Research Institute and, to some extent, Ontario Research Foundation. A.H. Ross and Associates has been involved with the project since its discovery in 1978.

The design is based on a throughput of 245 TPD of ore grading 1.9%U with an annual production rate of 1,500,000 kg of uranium. The process flowsheet is as shown in Figure 2.1.

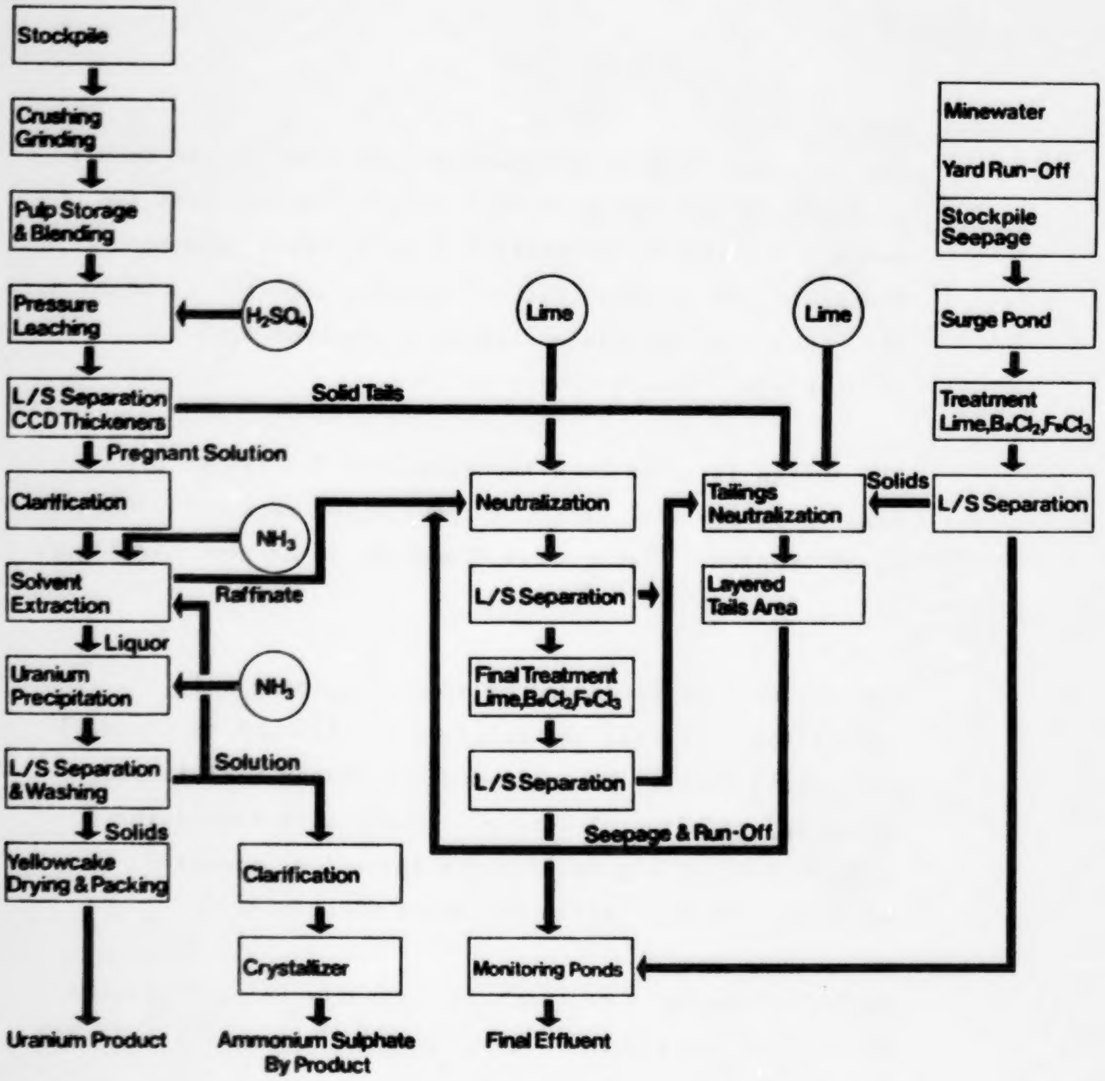
The ore will be recovered by open pit mining and stockpiled. It will be reclaimed and trucked to the mill on a daily basis. The main process stages will include crushing, grinding, leaching, liquid/solid separation, solvent extraction, uranium precipitation, ammonium sulphate crystallization and waste management.

Further laboratory testwork is currently being conducted. Mini pilot plant work on a 10 tonne bulk sample is planned for later this year.

2.2 Description

2.2.1 Stockpiling

The mining plans include mining the orebody and stockpiling it in a two year period with the mill start-up occurring three months after the start of stockpiling. Although the estimated average mill feed grade is 1.9%U and 3.32%As, analyses of drill cores indicate grades as high as 45%U and 63%As. Thus it is critical to obtain maximum blending of



Process Flowsheet
Midwest Project
Figure 2.1

the ore prior to leaching as the design limits are only 1.6%U to 2.2%U. This may be done by:

i) maximum blending during the construction of the ore stockpile, or by
ii) segregation of the ore into various small stockpiles according to grade (either U or As) and blending prior to feeding it to the mill.

Because of the large variations in grade, for design purposes, poor blending of the mill feed is assumed and further blending allowed for in the neutral pulp circuit.

2.2.2 Crushing

The ore from the stockpile will be discharged into a 50 tonne hopper and crushed to -200 mm with a jaw crusher. The ore will be fed directly to the mill from the crusher discharge.

2.2.3 Grinding

The crushed ore will be ground to -35 mesh (~30% minus 200 mesh) in a SAG mill in close-circuit with a screen classifier. A neutral thickener will be included to ensure pulp densities of 50% solids as leach feed.

The work index of the ore has been shown to vary from 9 to 26. To accommodate this variation the proposed mill design parameters will allow grinding of "normal" ore on day shift only, with harder or softer ores accommodated by varying the grinding time. The grindability of the ore will be further investigated on a small scale at Aerofall Mills Ltd. during the course of the pilot plant work.

The installation of a ball mill, in addition to the SAG mill, is being considered. This will be reviewed based on operating experience of other mills handling sandstone type ores.

2.2.4 Neutral Pulp Storage and Blending

Pulp from the neutral thickener is pumped to one of four neutral pulp storage pachucas according to the U grade i.e. each pachucas has a pre-determined limit. An on-stream analyzer determines the uranium content of the pulp and a micro-processor selects the storage pachuca accordingly. This circuit is shown in Figure 2.2.

Pulp is then withdrawn from each pachuca in weighted amounts according to grade in order to provide leach feed material assaying between 1.6%U to 2.2%U. This is also controlled by a microprocessor.

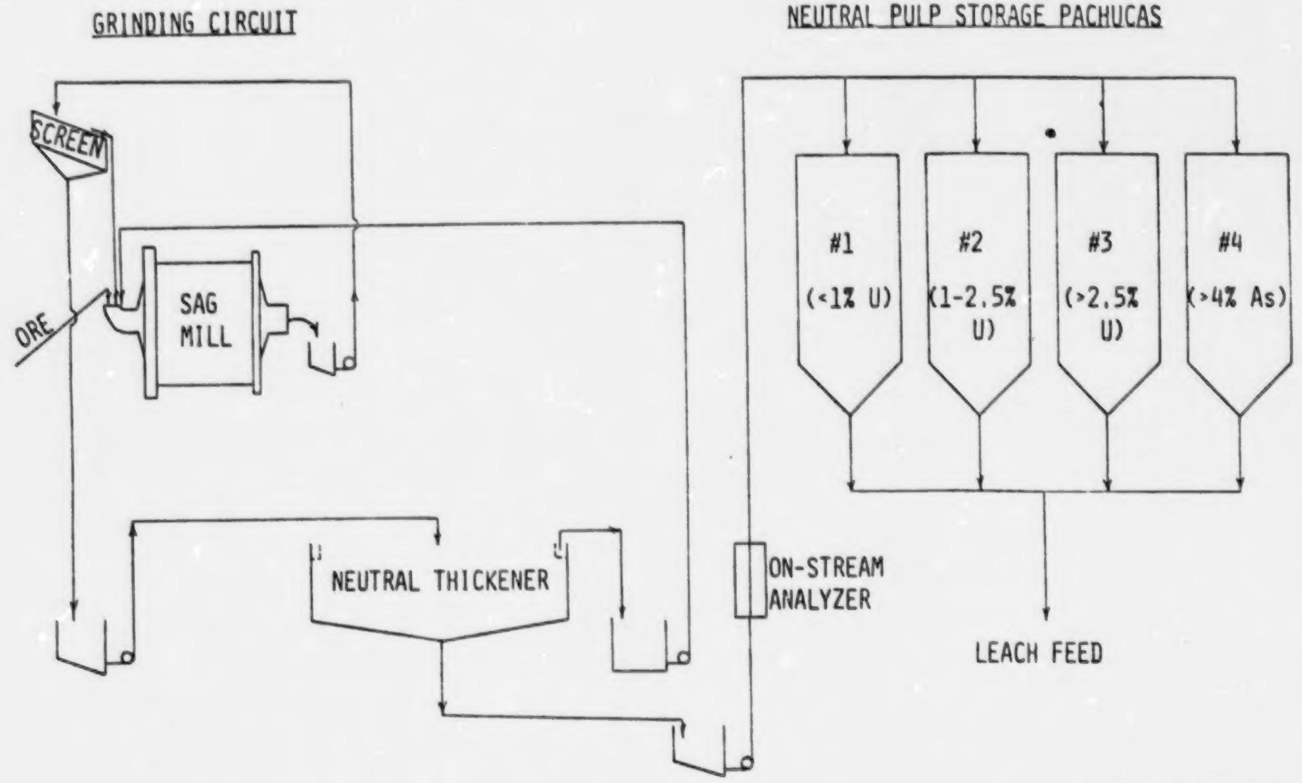
Typical ranges for the storage pachucas could be:

Neutral Pachuca No.	1	<1.0%U	<4% As
Neutral Pachuca No.	2	1.0-2.5%U	<4% As
Neutral Pachuca No.	3	>2.5%U	<4% As
Neutral Pachuca No.	4	>4%As	

The neutral pulp storage pachucas will be rubber lined in order to permit their possible usage as atmospheric leach vessels if careful control of the stockpile deposition and reclamation results in sufficient blending. This will eliminate the need for storing pulp in pachucas according to grade variation.

2.2.5 Leaching

Several methods of leaching were tested in the course of the process development. Alkaline leaching did not yield adequate extraction of



MIDWEST PROJECT
FIGURE 2.2
NEUTRAL PULP BLENDING

uranium and was therefore discontinued. Conventional atmospheric acid leaching in agitated tanks or pachucas will yield high extraction of uranium providing sufficient oxidant, eg. sodium chlorate, is available. However, this method of leaching gives variable uranium extraction because of the normal high arsenic content.

Leaching in autoclaves using sulphuric acid and oxygen is the proposed method for extracting the uranium. Testwork has shown that this method gives consistently high yields of uranium into the leach solution even when there is considerable variation of other major elements, including arsenic and nickel.

Typical pressure leaching data for varying leach conditions are given in Table 2.1 and Table 2.2.

TABLE 2.1

Typical Leach Data

		PRODUCTS	
HEAD GRADE	LEACH CONDITIONS	SOLUTION	RESIDUE
2.0%U	Temp = 70 -80°C	20 g U/L	0.02- 0.03%U
	Pulp = 50% w/w		
4.0%As	Time = 4 hours	20–25 g As/L	2%As
2.2%Ni	Acid = 110kg/tonne	10–15 g Ni/L	1%Ni
(-35 mesh)	O_2 gas at 500 kPa	20–40 g H_2SO_4/L	

TABLE 2.2

Typical Leach Data

HEAD GRADE	LEACH CONDITIONS	PRODUCTS	
		SOLUTION	RESIDUE
2.0% U	Temp = 100°C Pulp = 40% w/w	10-14g U/L	0.04-0.06%U
4.0% As	Time = 6 hours	0.5-1.5g As/L	4-4.5% As
2.2% Ni	Acid = 100-120 kg/tonne	10-12g Ni/L	<1% Ni
(-35 mesh)	O_2 gas at 750 kPa	10-20g H_2SO_4/L	

In order to maximize the uranium extraction the required leaching conditions are 80°C, 500 kPa, 100 kg/t acid and 50% solids for 4-6 hrs. Under these conditions 99% extractions is obtained.

2.2.6 Liquid/Solid Separation

The leach pulp obtained is viscous and the residue contains varying amounts of clays (sericitic materials) which are not easy to filter and only settle to a terminal density of approximately 40% to 45% w/w.

Envirotech Corporation has recommended an 8-stage C.C.D. circuit based on bench scale testwork. This work also included evaluations of drum filters and horizontal belt filters. Table 2.3 is a summary of the liquid/solid separation data supplied by Envirotech.

Table 2.3

Liquid/Solid Separation Alternatives

SYSTEM	NUMBER OF STAGES	WASH RATIO KG SOLN./KG SOLIDS	DESIGN DATA
C.C.D.	8	4.2 $\frac{\text{Preg kg}}{\text{*D.S. kg}} = 3.67$	0.45 m^2/T.D.D. 10% solids in feed 40% solids in under flow 99.9% U recovery 500 g polymer/tonne
Drum Filter	3	5.4 $\frac{\text{Preg kg}}{\text{D.S. kg}} = 5.88$	45.4 kg m^{-2}h^{-1} 950 g polymer/tonne 35% cake moisture 99.9% U recovery
Belt Filter	3	5.4 $\frac{\text{Preg kg}}{\text{D.S. kg}} = 5.93$	200 kg m^{-2}h^{-1} 500 g polymer/tonne 32% cake moisture 99.9% U recovery

*D.S. = dry solids

It is significant to note that approximately 50% of the total process acid requirement is used in leaching and 50% is used in the washing stages of the C.C.D. circuit.

This acid addition in the C.C.D. circuit is necessary to stabilize the leach solution and to inhibit the formation of precipitates which may form during washing. As a general rule, it is necessary to wash with a solution containing approximately the same concentration of free acid as is contained in the undiluted leach liquor at the completion of the leach stage.

After the C.C.D. stages, the pregnant solution is clarified through a filter and made suitable for treatment in the solvent extraction plant.

2.2.7 Solvent Extraction

The solvent used in the solvent extraction plant will consist of three components, typically mixed in the proportions shown in Table 2.4.

Table 2.4

Organic Make-up

Component	% by Volume	Specific Gravity
Tertiary amine	5	0.81
Decyl alcohol	5	0.84
Kerosene	90	0.82
Total	100	0.82

The solvent extraction plant will receive a clarified pregnant solution which will be fed at a predetermined, controlled rate. The analysis of a typical pregnant solution feed to extraction is given in Table 2.5.

Table 2.5

Typical Pregnant Solution Analysis

Component	Concentration g/L
U	4.19
As	5.41
Ni	2.85
Fe	1.60
Mo	0.08
Total SO_A^{-2}	42.5
Sol. SiO_2	0.23
Suspended soilds	0.020
free acid	28
emf.	-400 mv

Extraction will take place countercurrently in four stages of mixer-settlers with organic being the continuous phase yielding a raffinate containing <0.001 g U/L.

Metallurgical design criteria is given below in Table 2.6.

Table 2.6

Proposed Metallurgical Design Criteria : Extraction

Parameter	Units	Data
solvent loading	kg U/m^3	4.5
recycled solvent loading	kg U/m^3	0.005
raffinate analysis	kg U/m^3	0.001
mixer O/A ratio	--	1.3
mixer retention time	min	1 - 1.5
settler specific flow	m^3/h/m^2	3.0
after-settler specific flow	m^3/h/m^2	1.3

Extraction of both arsenic and molybdenum occur concurrently with the uranium extraction. However, the co-extracted arsenic and molybdenum levels in the loaded organic are generally low in relation to the uranium.

Typical arsenic and molybdenum loadings based on testwork are shown in Table 2.7.

Table 2.7

Arsenic/Molybdenum Organic Loadings

Solution	Analysis in g/L	
	As	Mo
Pregnant	5.0	0.5 - 0.1
Raffinate	4.0	.01
Loaded organic	.05 - .08	.05 - .08

The loaded organic phase coming from the No. 1 extraction mixer-settler contains in addition to uranium and molybdenum, impurities of arsenic, sulfuric acid, iron and silica. These must be removed before the organic phase reaches the stripping section. The solvent plant design incorporates the use of two dilute acid scrub stages, followed by a single water scrubbing stage.

Co-extracted arsenic is removed by dilute acid scrubbing of the organic. Two stages are sufficient to reduce the arsenic to an acceptable level. The water scrub stage removes entrained raffinate and acid scrub water in the organic.

The design criteria for the scrubbing section of the plant are shown in Table 2.8.

Table 2.8

Proposed Metallurgical Design Criteria: Scrubbing

Parameter	Units	Data
water scrub stages	---	1
H_2SO_4 scrub stages	---	2
acid strength	kg/m^3	10
mixer O/A ratio	---	1.3
mixer retention time	mins	4
settler specific flow	$m^3/h/m^2$	3.0

The solvent plant design will incorporate countercurrent pH controlled ammonium sulphate stripping to recover and concentrate the uranium from the loaded organic. The strip circuit has five stages of countercurrent mixer-settlers and a single stage after-settler. Anhydrous ammonia is used for

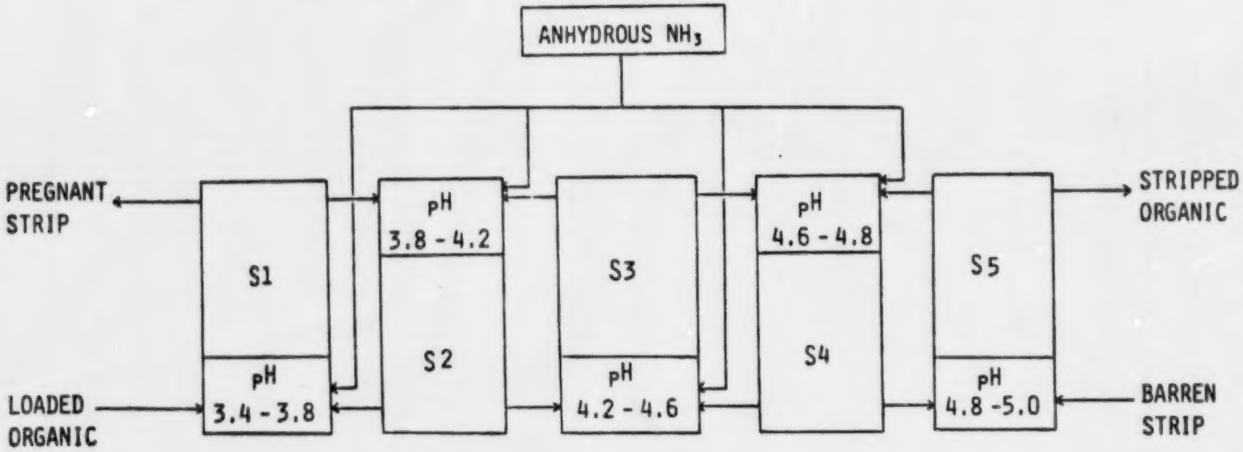


FIGURE 2.3 - URANIUM STRIP CIRCUIT

pH control in the mixers. It is anticipated that 99%-plus stripping will be attained. The pregnant strip solution will contain 15 -18 g U/L.

Since uranium and molybdenum will co-load onto the organic phase, they must be individually separated in the stripping stages in order that the molybdenum specification for yellowcake is met. To accomplish this, selective stripping of uranium by a pH controlled, ammonium sulphate strip will be done. The circuit configuration is shown in Figure 2.3. It is expected that this circuit will be effective in giving a good uranium/molybdenum separation. Metallurgical design criteria are given in Table 2.9.

Table 2.9

Proposed Metallurgical Design Criteria: Stripping

Parameter	Units	Data
ammonium sulphate strength	kg/m^3	180
pH range	--	3.4 - 4.8
ammonia consumption	kg/kgU	0.47
strip loading	kgU/m^3	17.5
barren strip loading	kgU/m^3	0.01
mixer O/A ratio	--	1.3
mixer retention time	min	4
settler specific flow	$m^3/h/m^2$	3.0
stripped solvent loading	kgU/m^3	0.01

To prevent a build-up of molybdenum in the stripped solvent, it is passed through a regeneration stage. This consists of a single mixer-settler where the solvent is contacted with a sodium carbonate solution. Typical analyses for molybdenum in the plant streams are shown in Table 2.10. Table 2.11 gives the design criteria for regeneration.

Table 2.10

Molybdenum Content of Solvent Plant Streams

100% Regeneration

Stream	Mo gpl
Pregnant	0.085
Raffinate	0.005
Loaded org.	0.056
Stripped org.	0.038
Strip conc.	0.033
Carb. regen.	0.404
Regen. org.	0.002

Table 2.11

Metallurgical Design Criteria : Regeneration

Parameter	Units	Data
Regeneration Solution	-	Na_2CO_3
Strength	kg/m^3	100
% Solvent Treated	%	100
Mixer O/A Ratio	-	1.3
Mixer Retention Time	mins	4
Settler Specific Flow	$m^3/h/m^2$	3.0
After-Settler Specific Flow	$m^3/h/m^2$	0.2

The proposed purification and recovery circuit is shown in Figure 2.4.

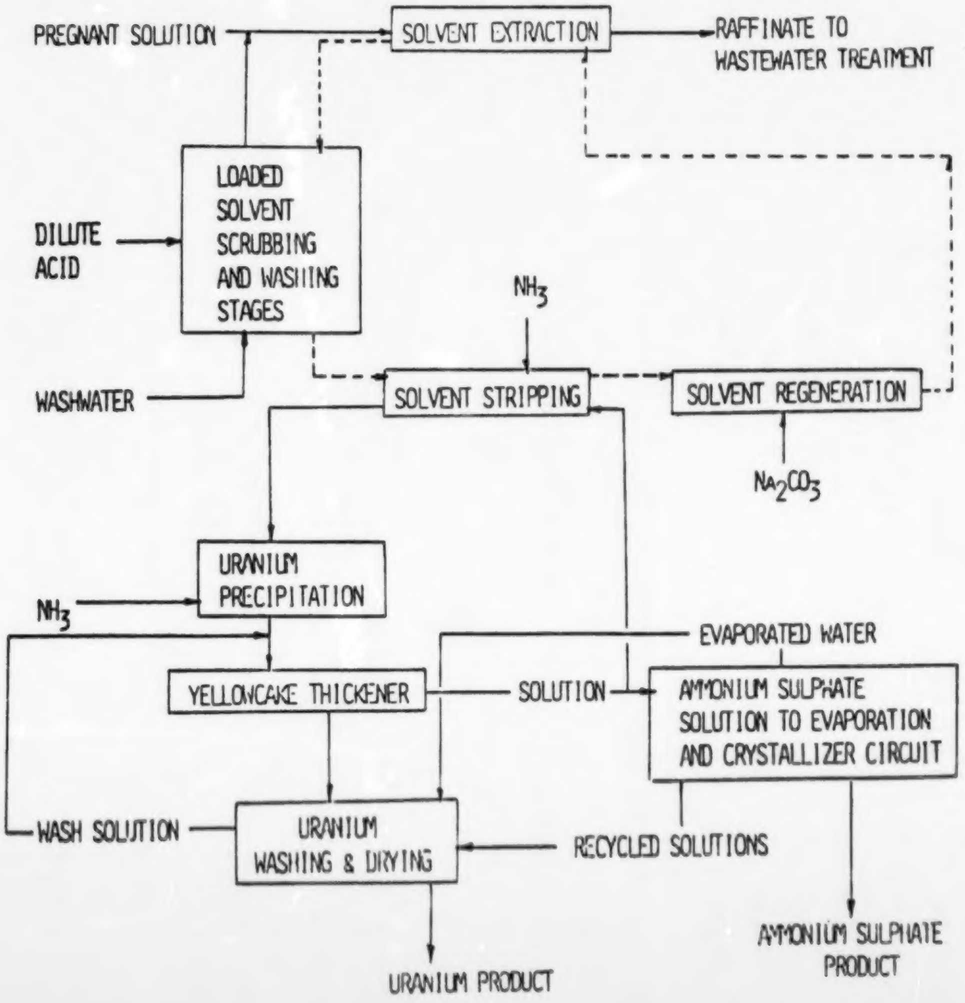


FIGURE 2.4 PURIFICATION AND RECOVERY OF URANIUM

2.2.8 Uranium Precipitation, Drying and Packing

Concentrated strip solution containing 15-18 g U/L is precipitated in two tanks in series. Anhydrous ammonia and air are added to the strip solution to bring the final pH to 7.0 -7.2. Two-hour retention time is provided in the precipitation tanks. The slurry is then settled in a thickener and the overflow barren strip solution is returned to the solvent stripping stages after dilution to reduce the ammonium sulphate level to the original 180 g/L. Excess solution is removed, filtered, and stored for further processing.

The precipitated yellowcake is thickened, reslurried with water and centrifuged. The cake is then fed to a multiple hearth drier. After drying, the yellowcake is cooled, crushed to minus 6 mm and stored in a bin from which it is packed in drums for shipping.

2.2.9 By-Product Recovery

The excess ammonium sulphate in the strip solution is be approximately two kilograms per kilogram Uranium produced. Testwork has shown that a good quality ammonium sulphate fertilizer can be made by removing the water to form a crystalline product. The process reduces concerns for ammonia content in the mill effluent.

Table 2.12 shows typical analysis of the by-product ammonium sulphate produced from a lab scale solvent extraction -precipitation circuit in which the solvent phase has been cycled approximately 80 times.

Table 2.12

Typical Ammonium Sulphate Analysis

Element	Concentration (ppm)
U	<0.2
As	<10
Ni	3-30
Pb	<10
Se	0.3-1.0
Zn	20
Hg	<0.2
Cd	<10
Co	<10
Ra	0.2-3.0 pCi/g

Nickel and cobalt are possible candidates for recovery. At the end of the solvent extraction system, the raffinate contains approximately 2-3 g Ni/L and 0.1-0.3 g Co/L, in addition to arsenic and a host of other impurities. The potential revenue from these metals will necessitate further study.

2.2.10 Reagents

Reagents used, along with approximate consumption rates, are given in Table 2.13:

Table 2.13

Reagent Consumption

REAGENT	CONSUMPTION		SUPPLY
	KG/T	T/DAY	
Sulphuric Acid	220	54	on site from elemental sulphur
Oxygen	45	11	on site production
Lime	193	47	ball mill
Ammonia	12	3	tanker
Barium Chloride	9.5	2.3	bulk
Ferric Sulphate (or Ferric Chloride)	24	5.8	bulk
Sodium Carbonate	20	4.8	bulk
Flocculant Percol 351	0.4	0.1	bags
Percol 726	0.2	0.05	bags
Diatomaceous Earth	6	1.4	bags
Organic-Kerosene	280L/day		drums
-Isodecanol	32 kg/day		drums
-Amine	24 kg/day		drums
Steel Balls	100 tonnes/year		

It should be noted that operating parameters have not been optimized and actual reagent consumption rates may be different from above.

3. Waste Management

3.1 Solution Types

All wastewater during operation of the mill will be integrated into the mill process and wastewater treatment systems. The sources of wastewater requiring treatment are:

(1) mine sump water
(2) seepage and runoff collected from tailings area
(3) mill site runoff
(4) ore storage runoff
(5) raffinate
(6) mine equipment wash

Figure 3.1 shows an outline of the water distribution.

3.2 Wastewater Treatment

The treatment of waste products is an integral part of both the milling process and the overall waste management facilities. Figure 3.2 shows the wastewater treatment system.

Waste products from the milling process will consist of washed leach residue from the CCD washing ciruit, and raffinate from the uranium solvent plant. In addition, the waste management facilities (layered tailings area) will provide seepage and run-off which will require treatment.

These waste products will be neutralized separately with lime slurry. The treated raffinate and layered tailings seepage and run-off will be settled, thickened and the gypsum sludge removed by filtering. The filter cake will be combined with the neutralized leach residue, and sufficient lime added to stabilize to a terminal pH. This mixture will then be placed in the layered tailings area.

FIGURE 3.1 GENERAL OUTLINE OF WATER FLOW

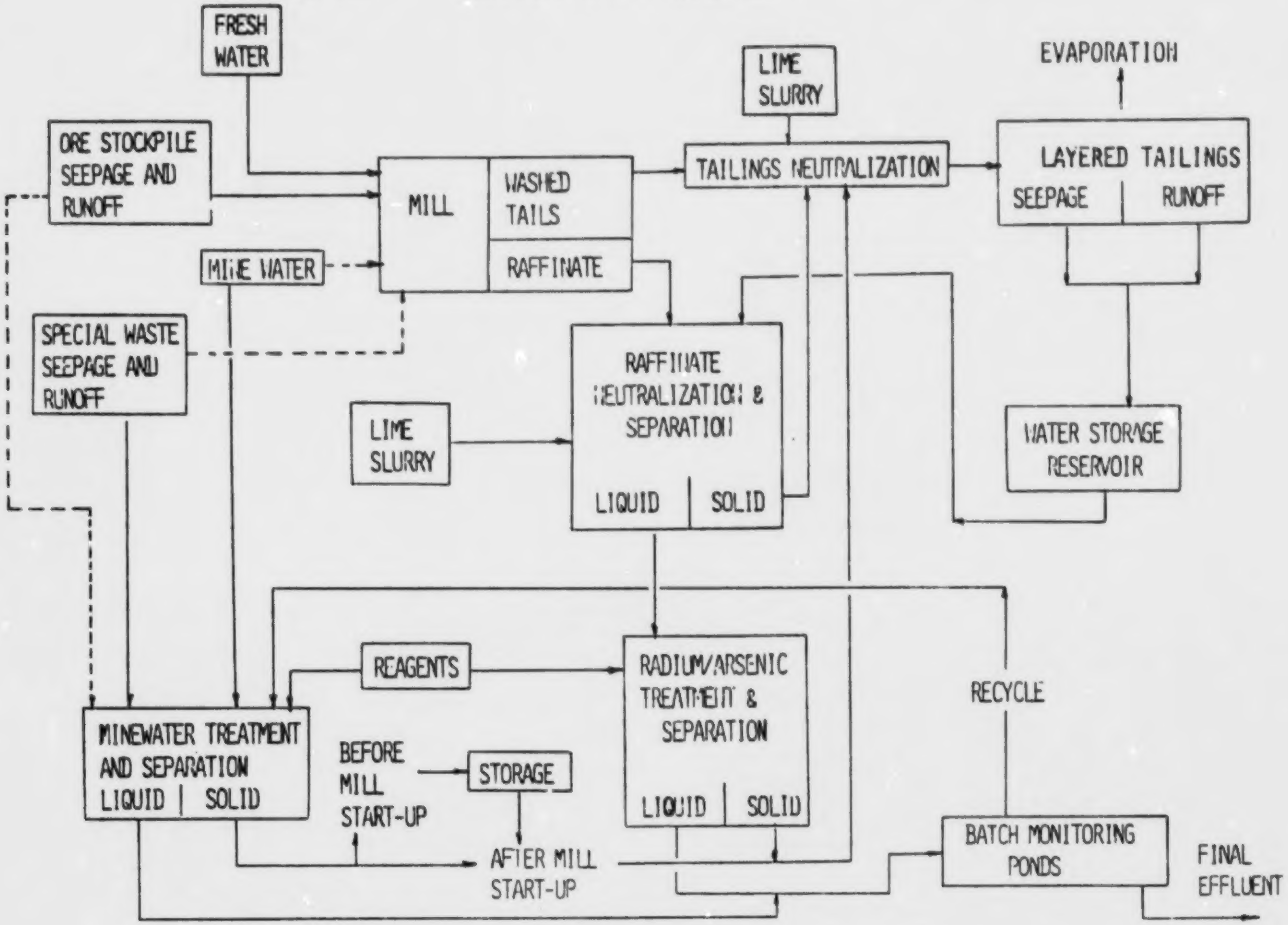
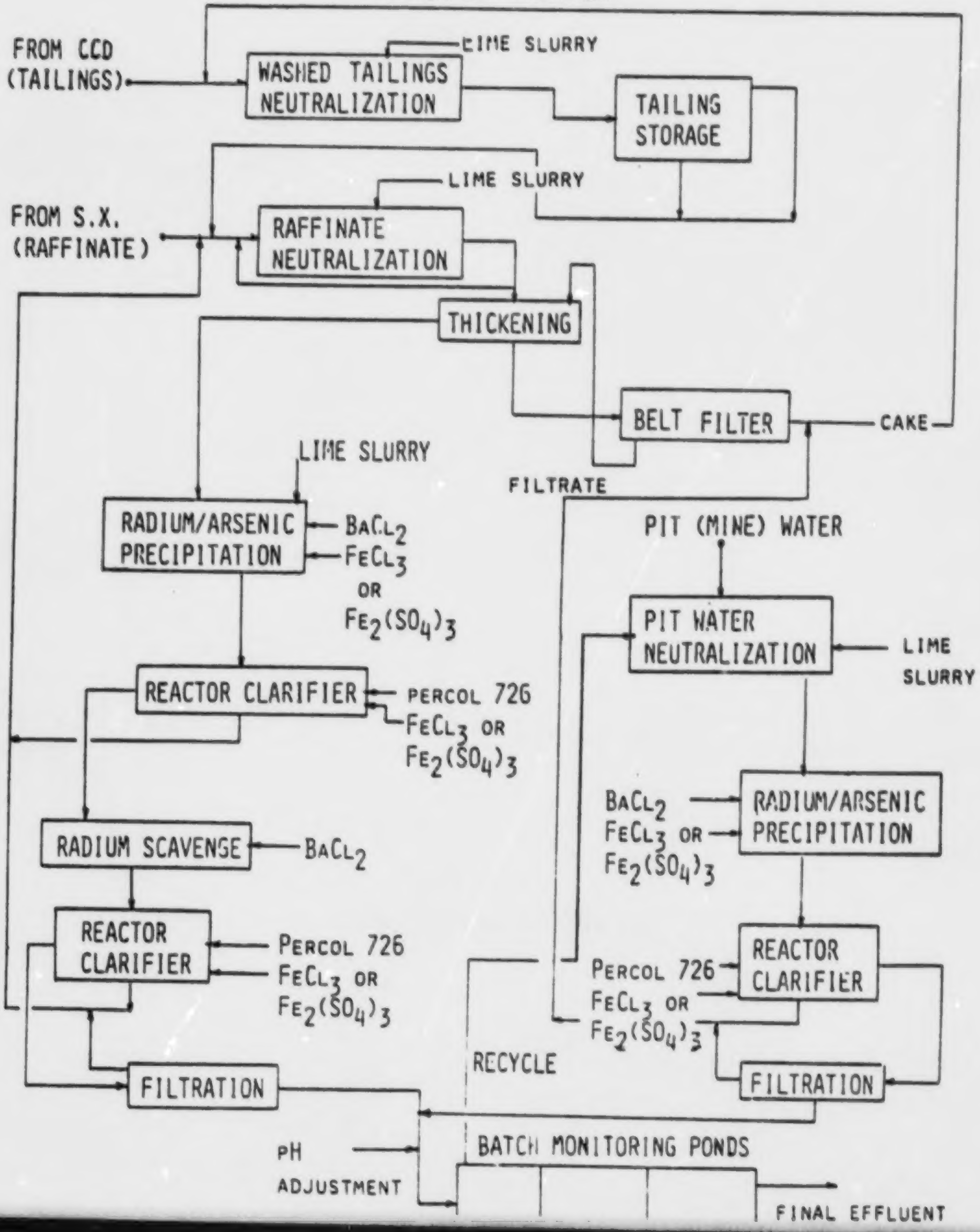


FIGURE 3.2

WASTEWATER TREATMENT



Solution which has been removed during the process of thickening and filtering of the gypsum precipitate must be further treated for residual levels of arsenic and radium. For this treatment both a secondary and tertiary stage is provided for in the design. The secondary treatment stage will involve the addition of ferric iron, barium chloride and, if required, lime slurry, to the solution. The resulting sulphate-arsenate precipitate sludge will be removed from the solution by means of a clarifier. The tertiary treatment stage will involve the addition of barium chloride and ferric chloride as a flocculant. This solution will be clarified to remove the radium precipitate. The solution overflow from this clarifier (effluent) will require polishing and final pH adjustment before discharge to the monitoring ponds.

The ore stockpile run-off will be collected in a surge pond along with the minewater.

Treatment of the wastewater from this surge pond will involve the neutralization of the water with lime, precipitation of radionuclides and arsenic with barium chloride and ferric iron, followed by a clarification stage. Final treatment of the effluent will be done with a polishing filter. The effluent will be discharged to a monitoring pond common to both the minewater and raffinate treatment circuits.

Effluents contained in the batch-operated monitoring ponds will be systematically sampled and analysis obtained so that final effluent quality is assured and is compatible with all regulatory requirements before any release is made. Storage of the effluent will be required for a time period determined by laboratory turnaround time.

In the event that the effluent quality of a certain batch is found to be greater than the discharge limits, provision is made for retreatment of the 'off-spec' batch through the mine water neutralization circuit. The quality of the expected final effluent, from laboratory wastewater testwork is shown in Table 2.14.

3.3 Waste Management Facilities

The waste management facilities at the Midwest Project have been designed for storage of the mill tailings and the special waste in adjoining areas.

The area is designed to isolate the materials stored and to intercept and recover any contaminants which may escape in the form of seepage. The consulting engineers for this facility are Knight and Piesold Ltd. of Vancouver.

The general arrangement of the waste management facilities is illustrated in Figure 3.3 and 3.4. An important feature in the design of the tailings storage facility is the method used to deposit the tailings.

The tailings will be deposited from the southern end of the tailings storage facility using the sub-aerial technique. In this technique the tailings slurry is systematically discharged from spray bars onto the upstream end of a tailings beach after which it flows over a section of the gently sloping beach to form a layer approximately 100 mm thick. After the layer has been deposited the discharge is moved to another section of beach and the freshly deposited slurry is left to settle and bleed. During this stage the coarser particles settle towards the bottom of the layer and the finer fractions concentrate at the surface to form a dense cohesive crust. The supernatant liquid flows down the sloping surface of the beach and, in this case, will be continuously drained away through a previous filter zone incorporated in the upstream portion of the confining embankment at the north end of the facility.

TABLE 2.14

TYPICAL FINAL EFFLUENT ANALYSIS

CONTAMINANT		RESULTS FROM SEVEN BENCH TESTS	
		RANGE	AVERAGE
Sulphate SO_4^{-2}		1440 - 3600	2111
Arsenic	(mg/L)	0.11 - 0.58	0.3
Copper	"	0.01 - 0.035	0.02
Cadmium	"	<0.005	<0.005
Lead	"	<0.05 - 0.3	0.1
Iron	"	0.04 - 0.1	0.06
Molybdenum	"	0.20 - 0.58	0.27
Nickel	"	0.005 - 0.07	0.047
Cobalt	"	<0.005	<0.005
Selenium	"		
Total P	"	0.16 - 0.56	0.27
Uranium	"	0.06 - 0.54	0.17
Vanadium	"	0.025 - 0.16	0.09
Zinc	"	<0.005 - 0.015	0.006
Mercury	(ug/L)	<0.05	<0.05
Ra^{-226}	(pCi/L)	2.48 - 11.0	5.8
Th^{-230}	"	0.28 - 5.5	1.65
Th^{-232}	"		
Pb^{-210}	"	1.65 - 162.3	25.9
Pb^{-210}	"	1.93 - 173.3	27.8
Total Suspended Solids (mg/L)			
Total Dissolved Solids (mg/L)		2440 - 2540	
pH		7.5 (process design pH)	

After the settling and bleeding stage has been completed the drained surface is left exposed to air dry until the saturation in the layer reaches a predetermined level, which in this case will be between 80 and 90 per cent. This reduction in moisture content to a partially saturated condition induces substantial negative pore pressures which shrink and consolidate the solid particles.

Tailings deposited in this manner have the following characteristics:

- the heavily laminated structure results in a relatively low coefficient of vertical permeability,
- all residual moisture in the partially saturated interstices is permanently retained by the capillary tension, and, unless the material is resaturated from some external source has no propensity to yield seepage,
- the consolidated material has considerable structural integrity.

The supernatant liquid which is drained from the downstream end of the beaches will be collected independently from the underseepage and recycled to the wastewater storage reservoirs for treatment prior to discharge to the environment.

The deposition strategy combined with the underseal and seepage collector system will ensure that the partially drained condition of the stored tailings will be maintained throughout the operating life of the mine and after decommissioning.

A permanent seal will be placed over the entire facility when the mine is decommissioned to ensure that the partially drained condition is maintained and to reduce emanation to regulatory levels.

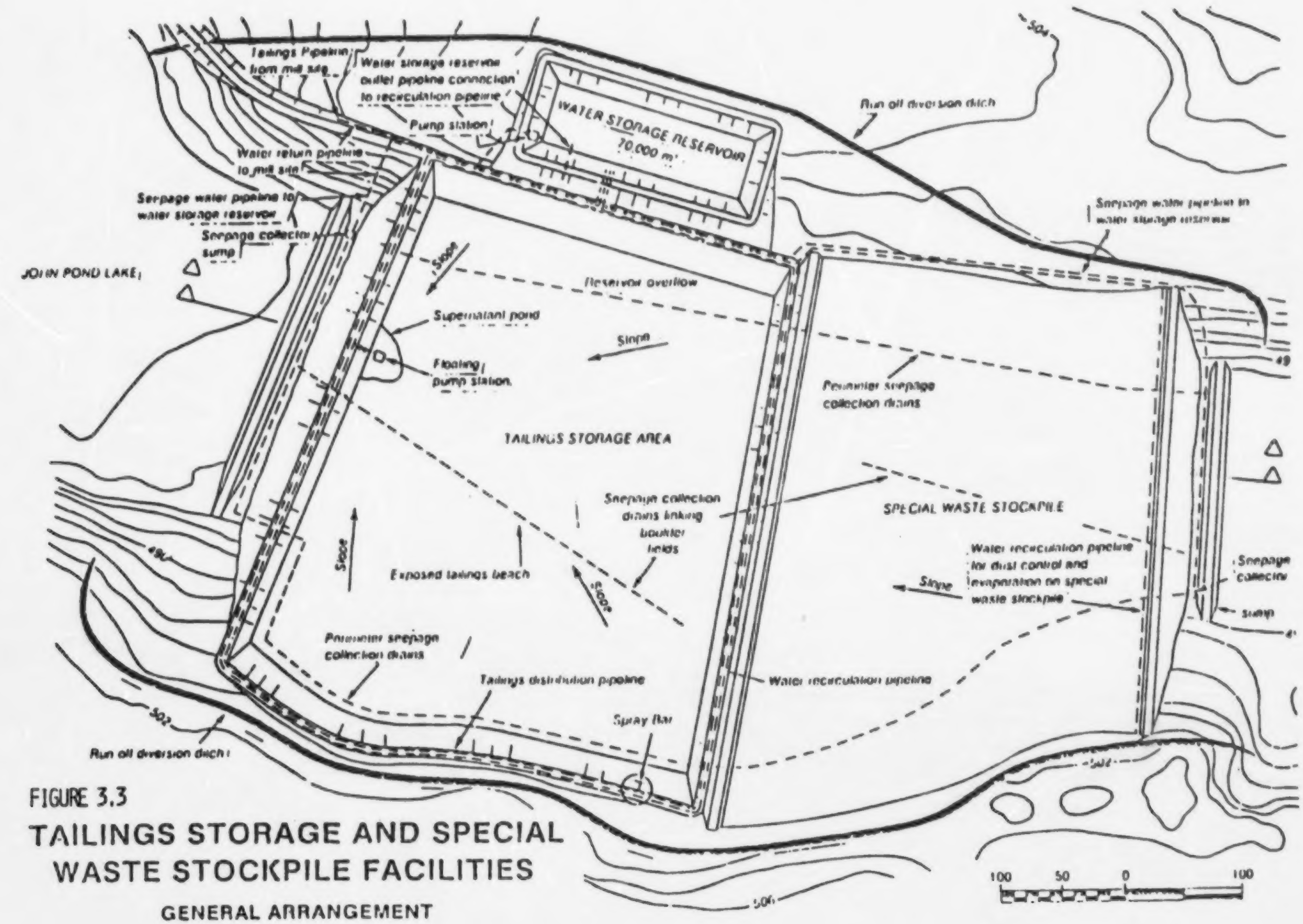
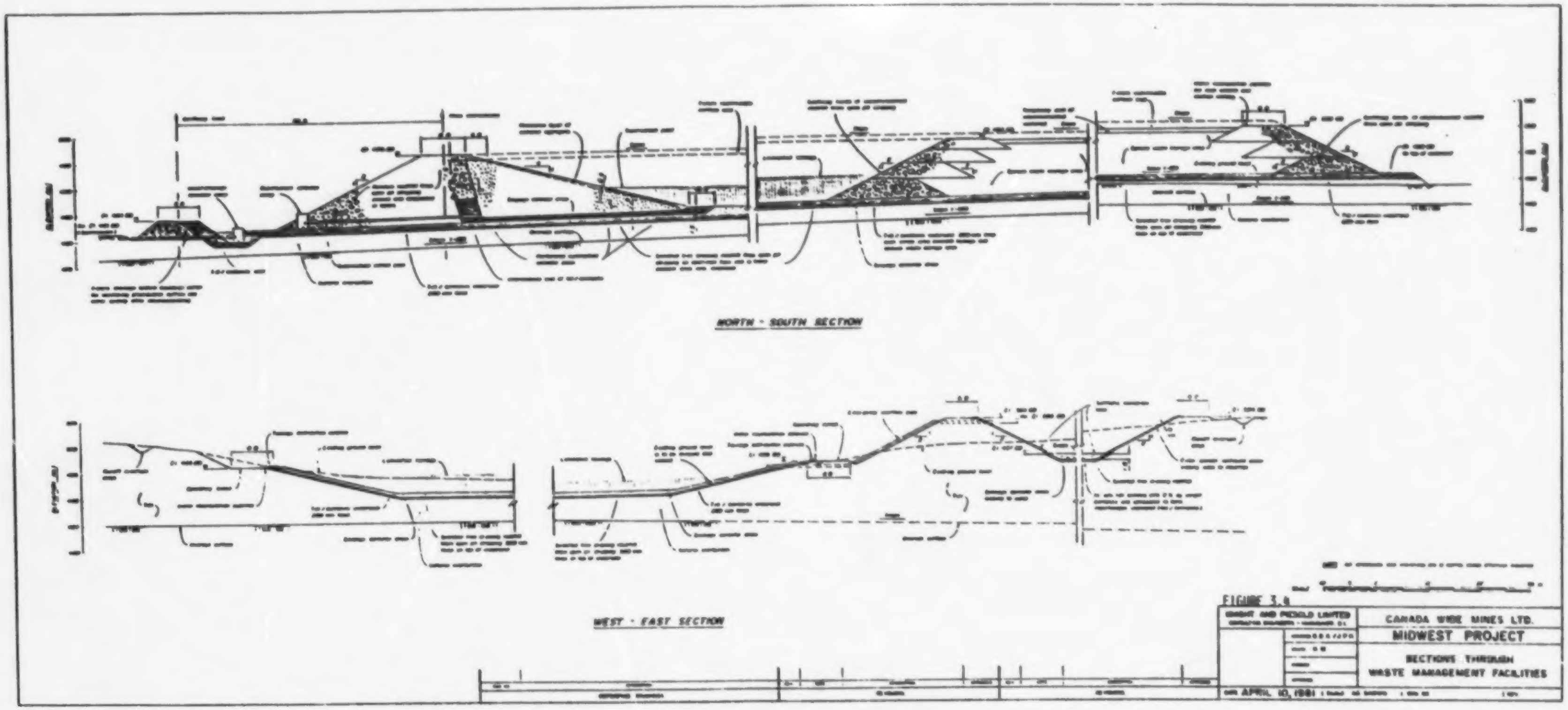

Tailings Pipeline from mill site
Water storage reservoir outlet pipeline connection to recirculation pipeline
Pump station
WATER STORAGE RESERVOIR 70,000 m³
Run off diversion ditch
504
Water return pipeline to mill site
Seepage water pipeline to water storage reservoir
Seepage collector sump
Seepage water pipeline to water storage reservoir
JOHN POND LAKE
Slope
Reservoir overflow
Supernatant pond
Slope
Floating pump station
Perimeter seepage collection drains
TAILINGS STORAGE AREA
Seepage collection drains linking boulder fields
SPECIAL WASTE STOCKPILE
Slope
Exposed tailings beach
Slope
Water recirculation pipeline for dust control and evaporation on special waste stockpile
Slope
Seepage collector
sump
Perimeter seepage collection drains
Tailings distribution pipeline
Water recirculation pipeline
Spray Bar
502
Run off diversion ditch
506
100 50 0 100

FIGURE 3.3
TAILINGS STORAGE AND SPECIAL WASTE STOCKPILE FACILITIES
GENERAL ARRANGEMENT



NORTH - SOUTH SECTION
WEST - EAST SECTION
KNIGHT AND PIESOLD LIMITED
CANADA WIDE MINES LTD.
MIDWEST PROJECT
SECTIONS THROUGH
WASTE MANAGEMENT FACILITIES
APRIL 10, 1981

FIGURE 3.4

DISCUSSION OF PAPER BY L. MELIS

E. Kraus	Would it be reasonable to return some of the raffinate to the CCD wash circuit?
L. Melis	Yes, recycling will be an important factor in the design. Some of the effluent will be considered for recycle.
M. Nevin	What is the pH of the $BaCl_2$ system?
L. Melis	There are 2 stages of addition of $BaCl_2$ with pH 7-8 in the final stage.
M. Nevin	What is the mine water quality expected and why treat it separately?
L. Melis	The laboratory work was done on a simulated basis. Acceptable levels of As (0.1-0.3 ppm As) and $<$10 pCi/L Ra were achieved. Also, the raffinate may be combined for treatment.
V. Lakshmanan	The arsenic will go into solution at the higher pH of neutralization. Will this be a problem?
L. Melis	Yes, that is why more than 1 point of control.
R. Weir	Caution should be noted on using simulated mine water.
I. Itzkovitch	Regarding Mo control in SX, how practical is it to control the circuit at 0.1-0.2 pH units.
L. Melis	Must maintain pH as low as possible for U-Mo separation.
I. Itzkovitch	Have you considered adding the $FeCl_3$ prior to leach?
L. Melis	We have considered precipitation of As in the leach, but there is poor L/S separation. Also a greater U loss.

G. Ritcey	What is the size of pilot plant and when is it being done?
L. Melis	10-12 tons.
I. Itzkovitch	Do you plan to have a silica removal stage prior to SX?
L. Melis	There is no problem in the present tests on Si - only clarification will be employed.
K. Wyper	What on-stream analysis are you going to use?
L. Melis	Possibly a G.E.
Question	What are you going to do with the Na_2CO_3 for regeneration?
L. Melis	Discharged to tails depending on U content
A. Moorish	Are there wide swings in acid consumption due to the different ore bodies?
L. Melis	No, the circuit design will allow up to 2.6 in grade, As/U of 2/1. Higher ratios not investigated. No wide swings in CO_3 minerals.

THE FUTURE OF THE RABBIT LAKE MILL OF GULF MINERALS CANADA LIMITED

A. Morrish P. Eng.
Project Manager
Mine Development
Gulf Minerals Canada Limited

MAY 1981

THE FUTURE OF THE RABBIT LAKE MILL OF GULF MINERALS CANADA LIMITED

THE PAST TEN YEARS

The Rabbit Lake mine and mill began producing uranium concentrates in 1975. At that time, the reserve had been estimated to contain 41 million pounds of U_3O_8 in approximately 6 million tons of ore.

At the end of 1980, approximately 4.0 million tons had been mined and 3.4 million tons had been milled. Clearly, then, the Rabbit Lake orebody is nearly exhausted and the operation is at an important crossroads in its history. In 1983 the last ton of rock will leave the Rabbit Lake pit. In 1985, the last pound of yellow cake will be drummed in the Rabbit Lake mill.

THE COLLINS BAY B-ZONE

The date for the exhaustion of the orebody had, of course, been defined at an early point in the project's history. The major objective of the Company's exploration programme in the 1970's was, therefore, to identify other orebodies so that Gulf Minerals Canada Limited could remain a major producer of uranium concentrates. The programme identified a number of prospects, some of which were in the immediate vicinity of the Rabbit Lake operation. One project, named the Collins Bay B-zone, was of special interest. It was located only six miles from the existing Rabbit Lake operation and, as definitive drilling progressed, it became clear that it was the prime target. A reserve containing 3.4 million tons of ore and 31.3 million pounds of uranium was eventually proven, in an orebody which is approximately 60 ft. below the surface.

The major advantages to the development of that orebody, therefore, were that its proximity to Rabbit Lake obviated the need to construct a new mill and auxiliary facilities, that it was relatively accessible and, as metallurgical testing eventually demonstrated, that with some modification its ore was amenable to beneficiation using the existing process.

MINERALOGICAL DIFFERENCES

The Rabbit Lake and Collins Bay B-zone orebodies have major mineralogical differences. A comparison of the analysis of typical samples of ore from Rabbit Lake and the Collins Bay B-zone is given in Table A.

Table A

A Comparison of Rabbit Lake and Collins Bay B-zone Ores

	% Contained			
	Rabbit Lake Typical Sample	Collins Bay B-zone High Grade	Collins Bay B-zone Average Grade	Collins Bay B-zone Low Grade
U_3O_8	0.31	15.0	0.62	0.28
Arsenic	0.003	10.2	0.47	0.49
Nickel	0.014	8.27	0.35	0.36
Molybdenum	0.002	0.12	0.01	0.015
Iron	-----	1.94	3.11	1.20
Vanadium	0.021	0.12	0.021	0.047
Thorium dioxide	-----	0.040	-----	0.003
Sulphur	0.10	0.56	0.07	0.10
Phosphate	0.16	-----	0.02	-----
Carbonate expressed as carbon dioxide	2.0 - 3.0	0.29	0.20	0.04

Metallurgically, the greatest difference is a higher concentration of nickel, molybdenum and arsenic bearing minerals in the Collins Bay B-zone ore.

Nickel and arsenic are present generally in combination as niccolite, a nickel arsenide mineral often found massive in combination with pitchblende. Although the sulphur content is low, minor sulphides have been recognized including millerite (nickel sulphide), galena (lead sulphide), chalcopyrite (copper iron sulphide), pyrite (iron sulphide) and ankerite (a dolomitic limestone, containing iron).

Gangue materials include sericite and kaolinite rich graphitic clays, chlorite, and carbonaceous material, generally in a host sandstone.

Uranium is present as uraninite, pitchblende and coffinite, the former often found massive or in fracture fillings, usually in association with niccolite.

From these and other analyses, there is an apparent, though not absolute relationship between uranium, nickel, and arsenic concentrations in the orebody.

PROCESS DESIGN OBJECTIVES

In designing modifications to the mill so that it can accept the Collins Bay B-zone ore, there are a number of environmental constraints, which affect the selection of the specific processes to be used.

The present Rabbit Lake mill consumes approximately 0.70 pounds of anhydrous ammonia for every pound of U_3O_8 produced. While some of the ammonia added is removed in the tailing neutralization and precipitation processes, a significant percentage reports with the effluent at the final point of discharge. Regulatory authorities have directed this Company to reduce ammonia levels in the effluent to a target level of 15 mg/l. This represents a reduction of present levels of the order of 95 to 97 percent. In addition, the significant quantities of arsenic and nickel in the Collins Bay B-zone ore require special consideration, to ensure that their concentrations in the final effluent are less than 0.5 mg/l in each case.

PROCESS SELECTION DESCRIPTION

Introduction

Early leaching tests confirmed that both arsenic and nickel would solubilize under leaching conditions which are favourable for uranium, and that extraction of both of these elements would be in excess of 80 percent. A series of physical tests was therefore conducted, to determine whether the minerals of nickel or arsenic could be effectively separated from the uranium minerals, before the leaching stage of the process. Gravity separation and flotation were both evaluated, but no reasonable degree of separation of arsenic and nickel from uranium was achieved. These techniques were therefore abandoned in favour of chemical removal of dissolved nickel and arsenic from effluent solutions, prior to discharge.

The process, as it is conceived today, is shown in Figure A.

Crushing and Grinding

The existing crushing and grinding circuit is satisfactory for Collins Bay B-zone ore. No changes are required to the equipment or equipment layout. The ore is basically a sandstone which is readily amenable to grinding. It is significantly easier to grind than the existing Rabbit Lake ore, having a Bond work index of 10.3 to 10.7, as compared to 14 to 16 for the Rabbit Lake ore.

Leaching

Test work has indicated that satisfactory extraction of uranium can be realized with the available retention time of 7.5 to 10.5 hours. Sulphuric acid consumption will be approximately 100 pounds per ton, or 50 to 60 percent of the Rabbit Lake ore's consumption. The difference in leaching conditions is the significant increase in sodium chlorate consumption to approximately 16 pounds per ton, or 300 percent of the present consumption. The projected average leaching conditions will therefore be:

- Grind	Nominal minus 35 mesh.
- Feed slurry density	50% solids by weight.
- Retention time	Six hours minimum.
- Leach temperature	50 to 55° C.
- Free acid	15 to 30 g/l.
- Terminal emf	Minus 400 to 450 millivolts.

Countercurrent Decantation and Classification

The countercurrent decantation and classification circuits will remain essentially unchanged. Although process economics may justify an additional seventh stage of C.C.D. thickening, that addition is not an essential modification to enhance the existing circuit's capability for processing the Collins Bay B-zone ore.

Solvent Extraction

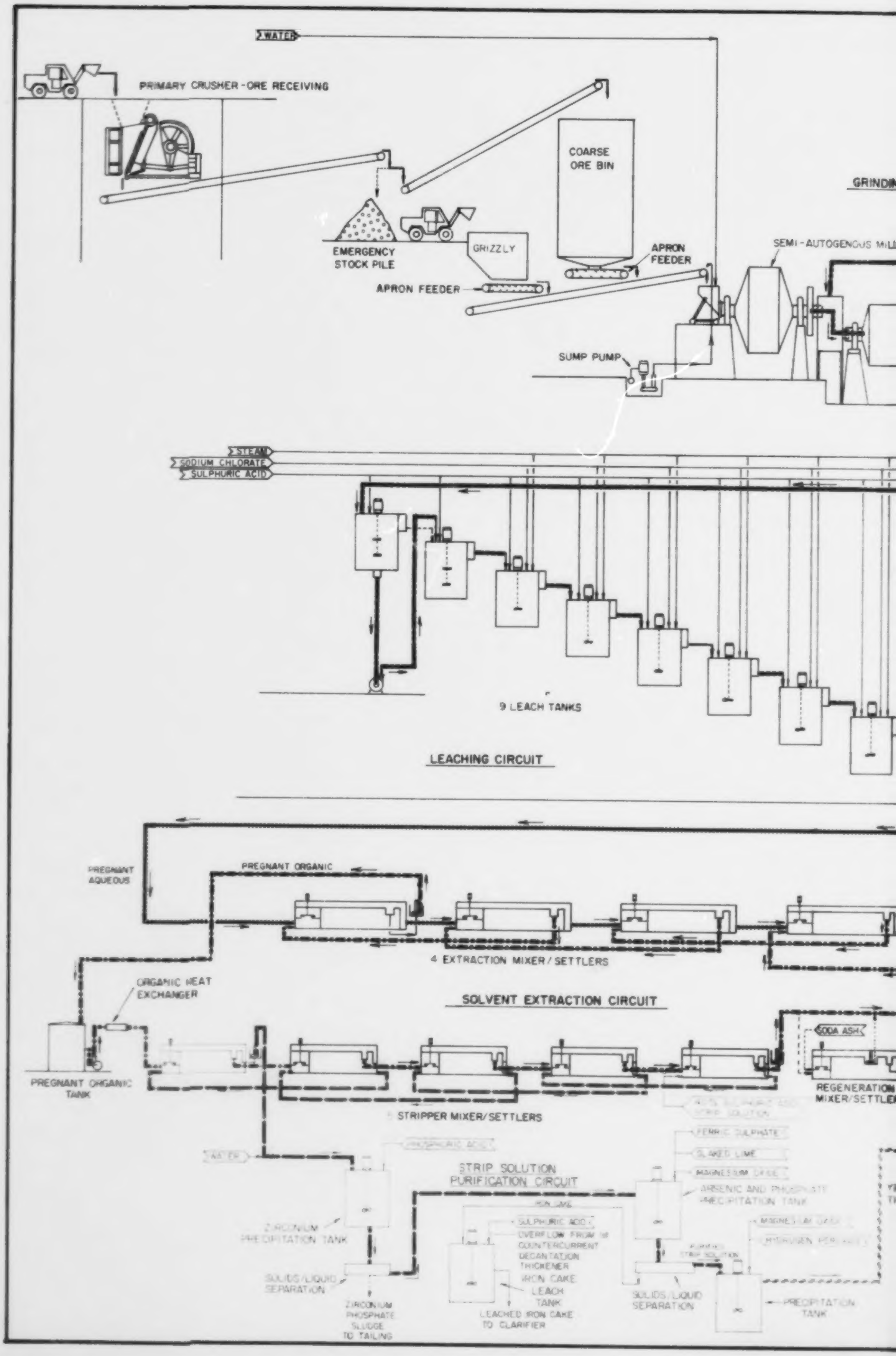
Initial test work with leach liquors from the Collins Bay B-zone ore was directed towards the application of the existing solvent extraction circuit, with ammonium sulphate stripping, to the purification and concentration of those liquors. Two major disadvantages were identified:

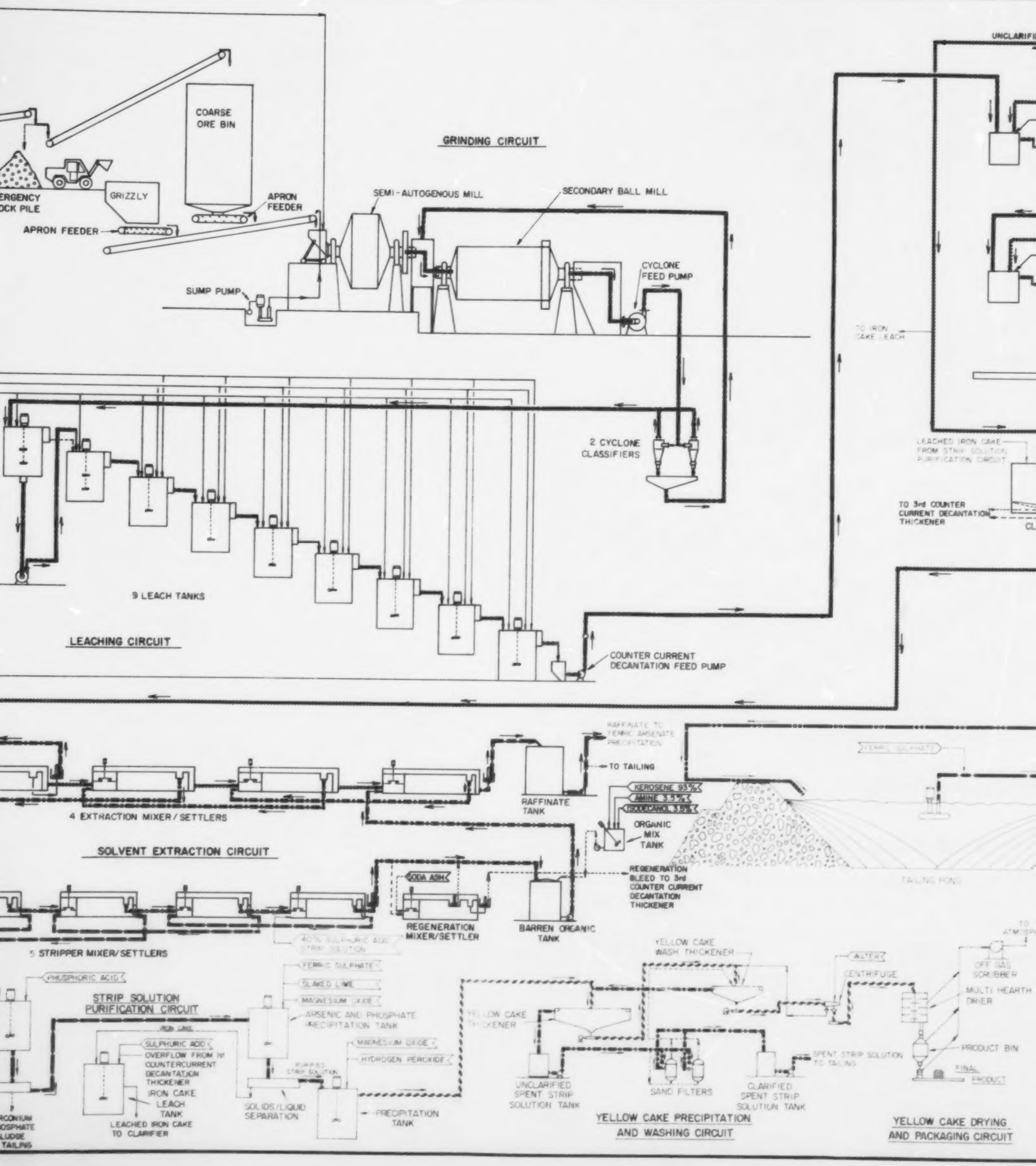
- Rates of sludge formation at the interface between the organic and aqueous phases in the stripping circuit were several orders of magnitude higher than presently experienced.
- To continue to use an amonium sulphate strip would involve the design and construction of a process for removing ammonia from the effluent, to achieve the environmental objective of less than 15 mg/l of ammonia in the effluent.

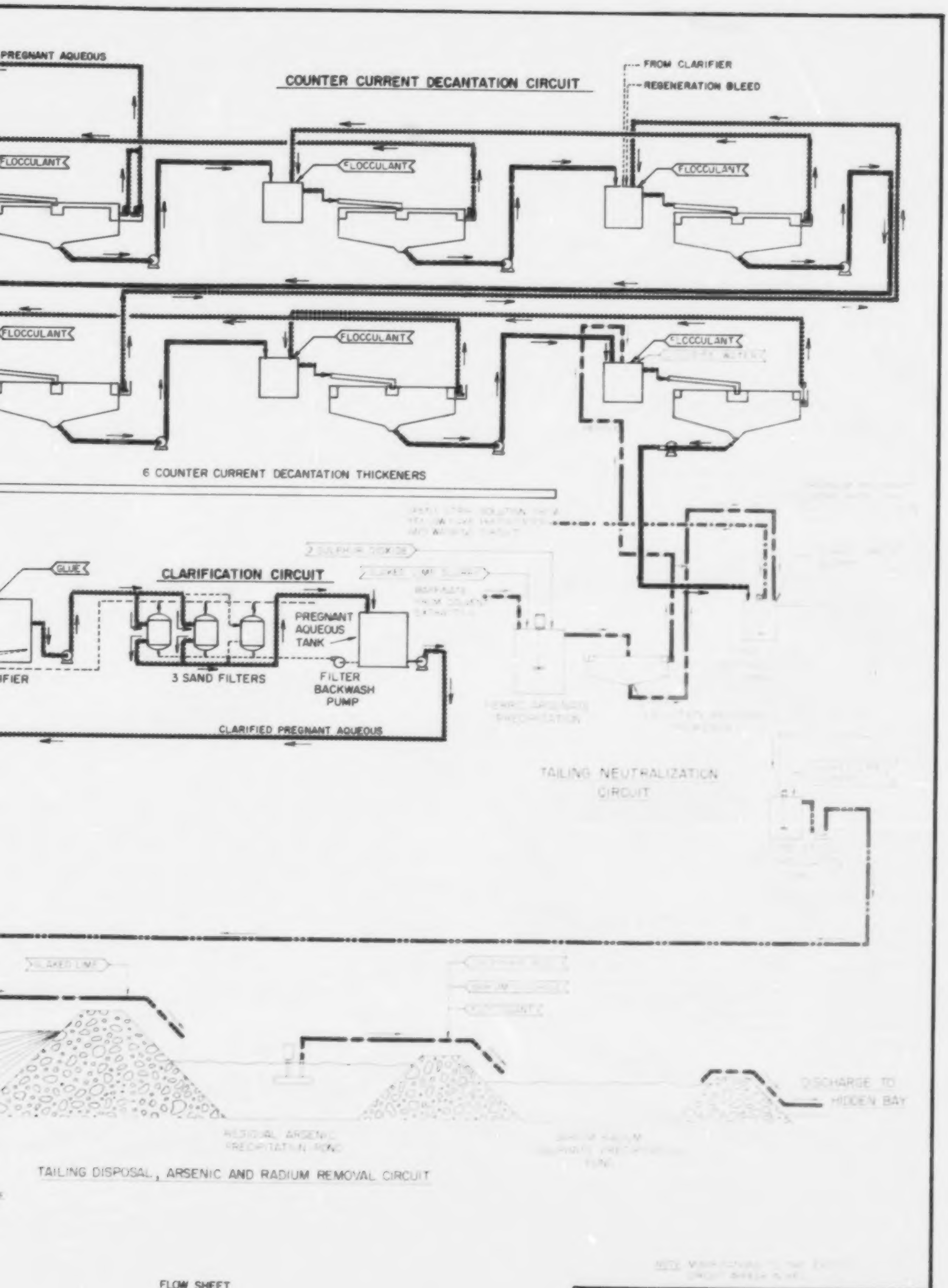
Sludge formation rates could be effectively reduced by pretreatment of the solvent extraction feed solution, but not eliminated entirely. In addition, although a process for removing ammonia from the effluent does exist, its ability to extract at the required efficiency is questionable. As a result, the existing tertiary amine, ammonium sulphate solvent extraction process was abandoned at an early stage.

Test work was then directed at a solvent extraction circuit which would use existing equipment and could be operated with an alternative stripping reagent. Sodium chloride and sodium carbonate were both evaluated and sodium chloride in particular, was found to be ideal.

Although sodium chloride stripped the uranium efficiently from the loaded organic solvent with a minimum rate of sludge formation, estimates indicated that the ammonia in the effluent would be replaced


WATER
PRIMARY CRUSHER - ORE RECEIVING
COARSE ORE BIN
GRINDIN
EMERGENCY STOCK PILE
GRIZZLY
APRON FEEDER
APRON FEEDER
SEMI - AUTOGENOUS MIL
SUMP PUMP
STEAM
SODIUM CHLORATE
SULPHURIC ACID
9 LEACH TANKS
LEACHING CIRCUIT
PREGNANT AQUEOUS
PREGNANT ORGANIC
4 EXTRACTION MIXER/SETTLERS
ORGANIC HEAT EXCHANGER
SOLVENT EXTRACTION CIRCUIT
SODA ASH
PREGNANT ORGANIC TANK
REGENERATION MIXER/SETTLE
5 STRIPPER MIXER/SETTLERS
40% SULPHURIC ACID STRIP SOLUTION
PHOSPHORIC ACID
WATER
FERRIC SULPHATE
SLAKED LIME
MAGNESIUM OXIDE
STRIP SOLUTION PURIFICATION CIRCUIT
ARSENIC AND PHOSPHATE PRECIPITATION TANK
IRON CAKE
SULPHURIC ACID
OVERFLOW FROM 1st COUNTERCURRENT DECANTATION THICKENER
ZIRCONIUM PRECIPITATION TANK
MAGNESIUM OXIDE
HYDROGEN PEROXIDE
PURIFIED STRIP SOLUTION
IRON CAKE LEACH TANK
SOLIDS/LIQUID SEPARATION
ZIRCONIUM PHOSPHATE SLUDGE TO TAILING
LEACHED IRON CAKE TO CLARIFIER
SOLIDS/LIQUID SEPARATION
PRECIPITATION TANK


ERGENCY
OCK PILE
GRIZZLY
COARSE
ORE BIN
APRON
FEEDER
APRON FEEDER
GRINDING CIRCUIT
SEMI-AUTOGENOUS MILL
SECONDARY BALL MILL
SUMP PUMP
CYCLONE
FEED PUMP
2 CYCLONE
CLASSIFIERS
9 LEACH TANKS
LEACHING CIRCUIT
COUNTER CURRENT
DECANTATION FEED PUMP
UNCLARIFI
TO IRON
CAKE LEACH
LEACHED IRON CAKE
FROM STRIP SOLUTION
PURIFICATION CIRCUIT
TO 3rd COUNTER
CURRENT DECANTATION
THICKENER
CL
4 EXTRACTION MIXER/SETTLERS
RAFFINATE
TANK
RAFFINATE TO
FERRIC ARSENATE
PRECIPITATION
TO TAILING
KEROSENE 93%
AMINE 3.5%
ISODECANOL 3.5%
ORGANIC
MIX
TANK
FERRIC SULPHATE
TAILING POND
SOLVENT EXTRACTION CIRCUIT
SODA ASH
REGENERATION
BLEED TO 3rd
COUNTER CURRENT
DECANTATION
THICKENER
REGENERATION
MIXER/SETTLER
BARREN ORGANIC
TANK
5 STRIPPER MIXER/SETTLERS
40% SULPHURIC ACID
STRIP SOLUTION
PHOSPHORIC ACID
STRIP SOLUTION
PURIFICATION CIRCUIT
FERRIC SULPHATE
SLAKED LIME
MAGNESIUM OXIDE
ARSENIC AND PHOSPHATE
PRECIPITATION TANK
IRON CAKE
SULPHURIC ACID
OVERFLOW FROM 1st
COUNTERCURRENT
DECANTATION
THICKENER
IRON CAKE
LEACH
TANK
LEACHED IRON CAKE
TO CLARIFIER
RCONIUM
OSPHATE
LUDGE
TAILING
PURIFIED
STRIP SOLUTION
MAGNESIUM OXIDE
HYDROGEN PEROXIDE
SOLIDS/LIQUID
SEPARATION
PRECIPITATION
TANK
YELLOW CAKE
WASH THICKENER
YELLOW CAKE
THICKENER
UNCLARIFIED
SPENT STRIP
SOLUTION TANK
SAND FILTERS
CLARIFIED
SPENT STRIP
SOLUTION TANK
SPENT STRIP SOLUTION
TO TAILING
WATER
CENTRIFUGE
YELLOW CAKE PRECIPITATION
AND WASHING CIRCUIT
TO
ATMOSP
OFF GAS
SCRUBBER
MULTI HEARTH
DRIER
PRODUCT BIN
FINAL
PRODUCT
YELLOW CAKE DRYING
AND PACKAGING CIRCUIT


PREGNANT AQUEOUS
COUNTER CURRENT DECANTATION CIRCUIT
FROM CLARIFIER
REGENERATION BLEED
FLOCCULANT
FLOCCULANT
FLOCCULANT
FLOCCULANT
FLOCCULANT
FLOCCULANT
6 COUNTER CURRENT DECANTATION THICKENERS
SPENT STRIP SOLUTION FROM YELLOW CAKE PRECIPITATION AND WASHING CIRCUIT
SULPHUR DIOXIDE
SLAKED LIME SLURRY
RAFFINATE FROM SOLVENT EXTRACTION
GLUE
CLARIFICATION CIRCUIT
PREGNANT AQUEOUS TANK
IFIER
3 SAND FILTERS
FILTER BACKWASH PUMP
CLARIFIED PREGNANT AQUEOUS
FERRIC ARSENATE PRECIPITATION
TAILING NEUTRALIZATION CIRCUIT
SLAKED LIME
RESIDUAL ARSENIC PRECIPITATION POND
BARIUM RADIUM SULPHATE PRECIPITATION POND
DISCHARGE TO HIDDEN BAY
TAILING DISPOSAL, ARSENIC AND RADIUM REMOVAL CIRCUIT
FLOW SHEET
SPENT STRIP SOLUTION
ORE
SLURRY
URANIUM BEARING AQUEOUS SOLUTIONS
BARREN AQUEOUS SOLUTIONS
ORGANIC SOLVENT
STRIP SOLUTION
YELLOW CAKE SLURRY
INTERMITTENT FLOWS

Gulf Minerals Canada Limited

MODIFIED FLOW SHEET OF RABBIT LAKE URANIUM MILLING PROCESS

DATE-OCTOBER 1980

FIGURE- *A*

by 12,800 pounds per day of sodium and 16,300 pounds per day of chloride originating from the stripping circuit. Sodium chloride was therefore environmentally unacceptable.

One solvent extraction process then remained, which would allow continued use of the existing process equipment. It would require a change from use of the tertiary amine presently employed, to a quaternary amine named Aliquat 336. The major advantage of the quaternary amine is that it may be stripped of its uranium using a sulphuric acid solution, thereby eliminating ammonia and avoiding the use of sodium chloride. Laboratory testing of the solvent again showed, however, that sludging at organic aqueous interfaces, in a modification of the Rabbit Lake solvent extraction plant, would be severe. Chemically, the amine was also less selective, with the result that arsenic in particular loaded onto the solvent and would have to be scrubbed in a significant plant addition. Both the kinetics of extraction and stripping were slower than those of the tertiary amine, so that additional solvent extraction cells would be required. While the quaternary amine therefore was environmentally acceptable, it was less than satisfactory as a possible alternative to the existing process.

One laboratory test work programme, in its initial stages, held some promise of success. It involved solvent extraction using the existing solvent, but substituted a 40% sulphuric acid solution for the present stripping reagent. Although initial results were encouraging, the approach was unconventional. Between June and October 1980, a considerable amount of test work was completed by Hazen Research Incorporated of Golden, Colorado, which eventually established the viability of the process. It is now intended to make the minor modifications which will be required to convert to a strong sulphuric acid stripping circuit.

The operating conditions for the extraction section of the solvent extraction ion exchange circuit will remain unchanged.

The sulphuric acid stripping solution will have a concentration of 400 g/l sulphuric acid and will be advanced through the stripping circuit at about 17 igpm, approximately one third of the flow rate required for ammonium sulphate stripping. Five mixer settler stripping cells are required, so that one extra cell must be constructed in the solvent extraction area.

Strip Solution Purification

The pregnant strip solution presently contains approximately 15 to 20 g/l of U_3O_8. Strong sulphuric acid stripping will produce a pregnant strip solution containing 100 to 125 g/l of U_3O_8. That solution will be diluted with water to approximately 25 g/l as feed to the strip solution purification circuit.

Table B shows a typical distribution of major dissolved ions throughout the various products of a continuous laboratory solvent extraction test. It demonstrates good selectivity for uranium but poor rejection of, in particular arsenic and molybdenum. If allowed to pass to the precipitation stage, they will precipitate with the uranium in quantities which will exceed the refiner's specifications. A strip solution purification stage is therefore included in the modified flow sheet. The first stage in the purification process will be the precipitation of zirconium as zirconium phosphate, by the addition of phosphoric acid to a concentration of 0.5 g/l. The zirconium phosphate precipitate will be filtered from the solution, washed and pumped to tailing neutralization.

The filtrate will then pass into agitated tanks where arsenic will be precipitated as ferric arsenate by the addition of ferric sulphate, followed by the addition of slaked lime to an approximate pH of 3.5 and magnesium hydroxide to an approximate pH of 3.8. The solution and precipitated solids (or iron cake) will pass to a thickener, from which the underflow solids will be removed. Since the iron cake will contain a minor amount of precipitated uranium, it will be leached with pregnant aqueous solution from the overflow of the first C.C.D. thickener, for uranium recovery. The leached iron cake will then be pumped to the clarifier, as shown on the flow diagram, so that the solubilized uranium may be recycled to the solvent extraction circuit for recovery. The ferric arsenate eventually leaves the circuit in the tailing stream from the C.C.D. circuit.

Product Precipitation

In the present circuit, anhydrous ammonia is used not only as a reagent for stripping but also for precipitation. In order to entirely eliminate its use in the Rabbit Lake mill, and to avoid precipitating molybdenum and silica, it will be replaced by the precipitant, hydrogen peroxide. Magnesium hydroxide will also be used to maintain the pH in the range 3.5 to 3.8 throughout the precipitation. Because a continuous hydrogen peroxide precipitation system has not been developed to date, the new system will be a batch process. When precipitation is complete, the solution and precipitated solids will be pumped to the yellow cake thickener, then washed and dried. The barren solution from the yellow cake thickener will be pumped through the existing sand filters for the recovery of any entrained product solids, and then to tailing neutralization.

Tailing Neutralization and Effluent Treatment

Almost five years of operation of the existing tailing disposal and effluent treatment at the Rabbit Lake mill has demonstrated its efficiency to produce an environmentally acceptable effluent in every respect except for its ammonia content. In the future, circuit modifications will eliminate the ammonia completely.

Two additional soluble contaminants must, however, be removed from the effluent when processing ore from the Collins Bay B-zone. They are arsenic and nickel and their maximum allowable concentrations in the final effluent are 0.5 mg/l in each case.

Arsenic may be precipitated from tailing slurries and from raffinate as either ferric arsenate or calcium arsenate. Tests completed to determine the potential for redissolution of each of those precipitates, have indicated that the former is less soluble than the latter. The modified process therefore, will precipitate the majority of the arsenic in that form.

The barren solution containing the majority of the arsenic will be treated separately. A one percent mixture of sulphur dioxide from the existing sulphuric acid manufacturing plant, in air, will be sparged into the agitated solution to ensure that all iron present in the solution is

Table B

Distribution of U, As, Mo, Zr, Si, and Bi in Alamine 336/H_2SO_4 Strip SX at 57 Organic Cycles - Collins Bay Liquor

	Assay, g/l							
	SX Feed	Raffinate	Pregnant Strip	Loaded Organic	Stripped Organic	% Extracted	% Stripped	Pregnant Strip Ratio Impurities/100 U
U_3O_8	*2.15*	*0.00069*	*112*	4.26	0.065	99.97	98.5	
As	1.78	1.79	0.35	±0.02	0.01	0.4	50	0.37
Mo	0.029	0.0056	0.68	0.14	0.087	81	38	0.71
Zr	0.0037	0.0017	0.059	±0.002	0.002	54	50	0.06
SiO_2	0.46	0.45	0.037	0.01	0.01	5	50	0.04
Bi	0.0032	0.0024	0.007		0.02			
H_2SO_4	10.0	12.0	293					
Cl	1.52	1.40	2.55					

oxidised to the ferric state. Lime must be added to maintain the pH at approximately 1.3. The pH will then be raised to 4.5 at which point the arsenic will precipitate as ferric arsenate. The resulting slurry will pass to a solution recovery thickener from where the solids will be pumped to a first stage neutralization circuit. The thickener overflow, if acceptable for reuse as wash in the C.C.D. circuit, will be recycled. Otherwise it also will be pumped to the first stage of the neutralization circuit.

In the first stage neutralization circuit, the discharge or discharges from the solution recovery thickener will be combined with the underflow tailing slurry from the last C.C.D. thickener, and the pH will be adjusted to 4.5 with slaked lime. The slurry will be held at that pH for approximately one hour to precipitate as much arsenic as possible as ferric arsenate. The slurry will then pass to the second stage neutralization circuit where the pH will be raised to 12.5 to 12.7 with slaked lime and retained for approximately two hours. There, the nickel will precipitate as nickel hydrate together with any residual arsenic.

Discharge from the second stage neutralization will be pumped to the tailing impoundment area.

SUMMARY AND ACKNOWLEDGEMENTS

The circuit modifications described will hopefully become reality before the Rabbit Lake mill runs out of ore in 1985. Negotiations with the regulatory authorities are ongoing to achieve that objective.

The design of the modifications is the result of almost two years of detailed test work performed in the laboratories of both Lakefield Research of Canada Limited and Hazen Research Incorporated, Golden, Colorado. The test work was supervised by R.C. Swider of A.H. Ross and Associates on behalf of Gulf Minerals Canada Limited.

DISCUSSION OF PAPER BY A. MOORISH

L. Melis	What is the As and Ra expected in effluent from Collins Bay?
A. Moorish	As 0.3 mg/L; Ra no difficulty expected but typically <10 pCi/L.
L. Melis	Have you considered the Knight and Piesold method?
A. Moorish	Yes we have considered it, but we are not dissatisfied with the performance of present waste management techniques.
L. Melis	Do you antiicpate any materials of construction problems with the 400 g/L H_2SO_4?
A. Moorish	No, tests indicate no change in performance.
I. Itzkovitch	Any phase disengagement and decomposition tests at higher temperature?
A. Moorish	No, we have not looked at the effect.
J. Fisher	What is the level of Cl^- concentration?
A. Moorish	?
J. Fisher	Are you required to maintain Cl^- levels?
A. Moorish	No, H_2O_2 is a possibility and high pressure leach, but high Cl^- levels could be a problem.
E. Krause	Would you choose IX over SX if you had to rebuild the Rabbit Lake plant?
A. Moorish	We would choose SX because of our background and confidence.
M. Witte	Are you going to line the tailings pit?
A. Moorish	The concept has not been defined but bentonite lining could be considered. Another possibility is a "halo" of contaminant around the pit.

G. Ritcey	Regarding crud, what is the basis for the tests on Collins Bay to go to the system chosen?
A. Moorish	They were based on qualitative tests comparisons. Sludge is due mainly (strip) to Zr and Bi. The use of activated C can decrease the amount generated.
A. Oliver	The Mo in the product is 0.8% on U basis. Do you anticipate problems in meeting specifications?
A. Moorish	The products meets Eldorado specifications.
V. Lakshmanan	Are you considering other than Kermac diluent?
A. Moorish	We also use Isopar M.

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Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

DEVELOPMENT OF AN ELUEX PROCESS FOR THE TREATMENT OF HIGH CHLORIDE URANIUM ELUATES

G.M. Ritcey, K.T. Price and E.W. Wong
Extractive Metallurgy Laboratory

February 1982

(Presented at the 1981 Annual CUPMC Meeting, Saskatoon, Saskatchewan, May 19-20, 1981 and for publication in the Proceedings)

Project ERP 4.4.0.2.01
Uranium Extraction - Conventional Technology

ENERGY RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 82-23 (OP&J) DRAFT

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by

G.M. Ritcey, K.T. Price and E.W. Wong

INTRODUCTION

In the processing of uranium ores by acid leaching, the usual approach to purification of the filtered and clarified leach solution is to use ion exchange or solvent extraction, or the Eluex process, a combination of both. Uranium mills in Canada, using either ion exchange or solvent extraction, recover uranium from the purified solution by neutralization precipitation with ammonia or magnesium oxide. The resultant "yellow cake", after drying, is normally sent to the Eldorado Nuclear Limited Refinery for further purification to UO_3 and subsequently UO_2. The process at the Eldorado Refinery involves digestion of the yellowcake in nitric acid and solvent extraction with tributyl phosphate. Occasionally certain yellowcake feeds present problems in the digestion stage or in the subsequent solvent extraction purification stage where a gel may occur after the leach or after mixing the leach with the solvent. Thus Eldorado Nuclear have developed an amenability test to evaluate mill yellowcake feeds prior to refining to ensure that the feed can be processed and whether it has to be blended into the circuit with other feeds.

One mill which has experienced difficulty in meeting the Eldorado amenability test for refining yellowcake feeds is the Madawaska Mines Limited at Bancroft, Ontario. The mill has tried various modifications, without success, in an attempt to prevent the gel formation on treatment of their feed at the refinery. The gel was caused by the presence of silica. As part of the Energy Program on Uranium Processing by conventional technology, CANMET undertook research on the problem after a request for assistance by the staff at Madawaska Mines. The feasibility of solvent extraction of the Madawaska chloride eluate was investigated as a method for uranium extraction and the production of a final product that would pass the yellowcake amenability test. The chloride eluate is the result of a brine elution of the uranium-loaded ion exchange columns.

PROCEDURE

The bench-scale solvent extraction tests, for the selection of the extractant and conditions for extraction, were performed in separatory funnels using a mechanical shaker. All tests were at ambient temperature. Typical analysis of the feed eluate is given in Table 1.

Table 1 - Typical analysis of eluate (g/L)

U	7.24	Si	0.052-0.15
Na	33.8	Mo	<0.0002
Ca	0.36	P	0.017
Mg	10.6	Th	0.092-0.11
Fe_T	0.069-0.17	S_T	28.3
Fe^{+2}	<0.001	Cl	30.2
		pH	1.2

The following reagents were evaluated in the test program, and the supplier is shown below:

Alamine 336	Henkel Corp.
Trioctylphosphine oxide (TOPO)	Cyanamid Canada
Trihexylphosphine oxide (THPO)	Cyanamid Canada
DEHPA	Mobil Chemical Corp.
TBP	Canada Colors
CES-429 (tri-n-octylamine)	Armak Chemicals Ltd.

The continuous solvent extraction testing was on a 10-stage unit purchased from Bell Engineering, Tucson, Arizona, having a mixer capacity of 620 ml.

The procedure that was used to produce "yellowcake" was that employed at Madawaska Mines, whereby the eluate (or the strip solution from solvent extraction) was adjusted to pH 7 over a 4-hour period by the addition of 10 w/o MgO. At pH 7, 2 ppm of NP-10 flocculant was added to aid settling. After settling, the barren solution was decanted and wash water added twice, followed by decantation. The yellowcake was then filtered and the product was dried at a temperature of 375-400°F.

All samples were analyzed, including the organic samples in order to arrive at a reasonable mass balance.

The amenability test, as used at Eldorado Nuclear, and provided by Madawaska Mines for evaluation of yellowcake products produced by our research, is as follows:

Eldorado Nuclear Ltd. Amenability Test for Chemical Precipitates

Scope

This test is an empirical one, designed to evaluate the extraction properties of chemical precipitates.

Outline

Chemical precipitate samples are digested in nitric acid and the leach liquor contacted with TBP/kerosene in a separatory funnel. The initial break and final separation times are recorded. The same solvent is contacted two more times with fresh leach liquor to saturate the solvent with uranium, and again initial break and final separation times are recorded after each shake-out.

Procedure

Add 240 ml of 5.5 N HNO_3 to a 1 litre tall beaker. Heat to 60-70°C. Add 100 g of sample to 10-20 g portions with stirring. The heat of reaction will bring the temperature near the desired 80°C. Digest for 30 minutes at 80°C with constant stirring. Cool to 40°C and dilute to approximately 240 ml with water and mix. Maintain a temperature of 40°C by keeping on hot plate. Add 80 ml of the 40°C leach liquor to 80 ml of 25% TBP in kerosene in a 250-ml separatory funnel. Mix for 3 minutes. Immediately observe and record the times for the initial break and final separation of the phases. Note and record the time of multiple phase breaks and the initial joining of the two phases. The explanation of the break times are given in Appendix I.

RESULTS

A limited number of exploratory tests were performed on the initial sample of ion exchange eluate received from Madawaska. Direct precipitation tests were performed using MgO received from Madawaska (containing 0.28% Si) as well as with reagent grade MgO (containing <.02% Si). Activated carbon was also investigated as a means of purifying the eluate, as well as two-stage precipitation tests (first stage to remove impurities at pH 3.5 followed by uranium precipitation at pH 7) were performed. The final yellowcake products from the tests were subsequently subjected to the evaluation procedure of HNO_3 dissolution and TBP extraction. There was no significant difference in the use of the different samples of MgO, with gels forming in the direct precipitation route. Although the two-stage precipitation resulted in an improvement in emulsion reduction, nevertheless a gel was formed.

Because of these initial tests, the subsequent work was concerned with the application of solvent extraction to process the eluate, that is the development of an eluex process.

Concurrent work, on the precipitation of uranium from the eluate by use of hydrogen peroxide, was successful in producing the desirable yellowcake, and will be reported in a subsequent paper.

SCREENING OF POSSIBLE EXTRACTANTS

The initial tests on the possible use of a tertiary amine, Alamine 336, for the extraction of uranium directly from the eluate proved unsuccessful, due to the relatively high chloride concentration. Therefore a series of bench-scale tests were performed on possible reagents for extraction and ease of stripping. The results of the screening tests are shown in Table 2. All solvents, in Shell 140 diluent, were acid pre-equilibrated at O/A 2/1 with 5% H_2SO_4 and the extraction was at room temperature at the conditions noted in Table 2.

Table 2 - Screening of possible extractants

Extractant	Eluate feed g U/L	O/A ratio	Time min	U ext'd (%)	Analyses of solvent g/L U	Th	Fe	Remarks
5% Alamine 336 + 5% Isodecanol	7.0	2/1	3	17.3	0.55	<0.001	0.002	
5% TOPO	7.0	2/1	3	27.0	1.36	<0.001	0.002	Emulsion at interface
5% CES-429	6.35	2/1	3	38.1	1.21	<0.001	0.031	Third phase
5% THPO	6.35	2/1	3	48.8	1.78	<0.001	0.001	
3.5 Alamine + 3.5% CES-429	6.35	2/1	3	49.8	1.58	<0.001	0.028	
10% Alamine + 5% TBP	7.08	2/1	3	49.9	1.80	<0.001	0.001	Third phase
75% TBP	8.56	2/1	3	67.0	2.42		0.0009	
3.5% CES-429 + 3.5% TOPO	6.35	2/1	3	68.0	2.16	<0.001	0.034	
3.5% Alamine + 3.5% TOPO	7.0	2/1	3	82.6	3.01	0.001	0.049	
3.5% Alamine + 3.5% THPO	6.35	2/1	3	89.8	3.08	<0.001	0.043	
3.3% DEHPA + 5% TBP	7.3	1.5/1	2	95.5	4.60	0.057	0.010	
3.3% DEHPA + 5% TBP	7.3	1.5/1	1	89.5	4.31	0.058	0.009	
3.3% DEHPA + 5% TBP	7.3	1.5/1	0.5	71.2	3.43	0.052	0.007	

The results of the screening tests indicated that the direct extraction of uranium from the eluate with a tertiary amine was not possible, due probably to the relatively high chloride content. However, the synergistic mixtures of Alamine 336 + TOPO or Alamine + Tri-H-P-oxide as well as DEHPA were capable of direct extraction of the uranium, although the latter also extracted any thorium and iron present. The use of TOPO with the alamine also extracted considerable iron. DEHPA appeared to offer excellent kinetics of extraction compared to the other reagents tested.

As a result of those tests the next series of tests were conducted on the Alamine-TOPO mixture, DEHPA-TBP and TBP. The first two extractants were evaluated for their ease of stripping the uranium off the solvent, while the TBP was considered for possible use if additional chloride were provided to the eluate. The following describes the extraction and stripping characteristics of these solvent extraction mixtures.

ALAMINE 336 + TOPO

Because the Alamine-TOPO mixture resulted in reasonable uranium extraction, stripping tests were then carried out to determine which reagent would be preferable. The tests were done at an O/A ratio 1/1, in contact with a loaded solvent containing 3.23 g U/L. The contact time was for 1 minute. The results are given in Table 3.

Table 3 - Stripping tests on Alamine-TOPO solvent mixture

Strip reagent	U stripped	pH		Remarks
	(%)	Init.	Final	
50 g/L NaCl	4.95	5.97	5.97	
130 g/L NH_4Cl	9.59	4.66	2.15	
50 g/L Na_2SO_4	10.5	2.2	2.2	
50 g/L Na_2SO_4	46.7	7.1	4.0	
130 g/L $(NH_4)_2CO_3$	57.8	7.6	7.7	Emulsion in organic phase
130 g/L $(NH_4)_2SO_4$	59.1	5.6	4.1	Emulsion and some crud
2.5% Sod oxalate	82.2		5.1	Good phase separation
50 g/L Na_2CO_3	98.5	11.0	10.1	Organic phase milky even after filtration

Although the Na_2CO_3 resulted in the most effective stripping, because of the cloudy nature of the organic the subsequent stripping was carried out with the sodium oxalate solution. First, the loaded solvent was scrubbed twice, O/A 1/2, with water prior to stripping with 2.5 w/o sodium oxalate solution at O/A 1.5. A strip solution, containing 7.08 g U/L was adjusted to pH 12.75 with H_2O_2. The product had the following analyses, in %: U 55, Fe 0.02, Mg 0.12, Si 0.21. The subsequent amenability test was performed with no gel being produced, indicating a probable acceptable refinery feed yellowcake. However, because the uranium grade was low and because the difficulty in stripping, required large volumes of strip solution, no further work was carried out.

DEHPA-TBP

Because of the relatively good extraction achieved with DEHPA, further tests were carried out on stripping the uranium from the loaded solvent. In the tests, a loaded solvent (3.3% DEHPA + 5% TBP in Shell 140) containing 4.38 g U/L was contacted with the aqueous strip solution for 3 minutes at an O/A ratio of 2/1. The results in Table 4 indicate that either a carbonate or bicarbonate strip are suitable for removing uranium. The bicarbonate strip is possibly preferable in view of the fact that the strip solution product was clear.

Table 4 - Stripping of DEHPA-TBP

Stripping reagent	U stripped (%)	Loaded strip (g/L) U	Th	Fe	Remarks
130 g/L NH_4HCO_3	99.8	6.66	0.036	0.013	Strip clear; organic slightly hazy
50 g/L Na_2CO_3	98.4	7.79	0.017	0.004	Strip clear; organic slightly hazy
270 g/L H_2SO_4	45.2	2.54	0.091	0.017	

USE OF TBP

In a series of tests on TBP, the feed eluate was varied in chloride content, from the initial 0.85 M NaCl up to 3.35 M NaCl. No additional acid was added. The 75% TBP, in Solvesso 150 diluent, was pre-equilibrated with 50 g HCl/L at O/A 5/1 prior to contact with the eluate for 3 minutes at O/A 2/1. The results are given in Table 5.

Table 5 - Effect of chloride concentration on extraction with TBP

Feed (M NaCl)	Eluate U g/L	Ext'n (%) U
0.85	8.56	67
1.35	8.56	69
3.35	8.56	83

The results indicated that if chloride were added to the eluate feed, then the solvating mechanism of extraction could be improved to 83% in one stage at a NaCl concentration of 3-3.5 M. Stripping was accomplished with water or dilute acid.

EFFECT OF WATER DILUTION OF ELUATE ON EXTRACTION WITH ALAMINE 336

A series of tests was carried out to determine whether dilution of the eluate with water prior to solvent extraction with a tertiary amine would result in improved uranium extraction by dilution of the chloride concentration. The tests were on an eluate containing 7.08 g U/L, and after water dilution the resultant solution was contacted at a phase ratio based on the original eluate volume, O/A 3/1. The results in Table 6 indicate that a water dilution of twice the eluate volume provides for increased uranium extraction. The tests were single-stage bench shake-outs.

Table 6 - Effect of dilution of eluate on uranium extraction

Eluate/H_2O ratio (ml)	U extracted (%)
no dilution	17.3
100/50	46.1
100/75	55.1
100/100	60.8
100/200	69.1

MIXER SETTLER TESTS

ALAMINE 336 CIRCUIT

Because the bench tests on the eluate in contact with the tertiary amine indicated the necessity for dilution due to the chloride ion present, confirmatory tests were run on a 4-stage mixer settler. The eluate was diluted with water over a range of eluate/water ratio of 1/0.75-1/2 and contacted with a solvent consisting of 5% Alamine 336 + 5% isodecanol in Shell 140 kerosene previously equilibrated with 5% H_2SO_4. Four separate runs were made at different dilutions, while maintaining the same equivalent ratio of O/A 2/1 relevant to the original eluate before dilution. The equilibrium pH varied between 1.2 to 1.8 with dilution from 1/0.75 to 1/2. The results of these continuous tests are shown in Fig. 1, indicating a dilution of 1/2 (eluate/water) provides for maximum uranium recovery.

Next, a circuit was run on the same solvent, employing 4 stages of extraction, 3 stages of stripping with 5% Na_2CO_3 and 1 stage of acid equilibration of the solvent with 5% H_2SO_4. Sodium carbonate was used as the stripping agent rather than ammonium sulphate because of the ease of pH control compared to the latter. Circuits were run with and without incorporation of a water scrub stage after extraction. The eluate feed, containing 7.24 g U/L, was diluted 1/2 (eluate/water) before extraction, resulting in a feed to solvent extraction containing 2.3 g U/L, 0.035 g Th/L, 0.027 g SiO_2/L, 0.028 g Fe/L, 4.12 g Mg/L and 10.8 g Cl^-/L. The O/A ratio for extraction was 1/1, using 180 ml/min of each phase, with the contact time in each mixer being 1 minute. However, only 93.5% extraction of uranium was achieved, as the phase ratio should have been 1.2/1 O/A. The loaded solvent was stripped in 3 stages with 5% Na_2CO_3 solution at an O/A ratio of 4/1. Prior to recycling the stripped solvent to extraction, the solvent was pre-equilibrated with 5% H_2SO_4 at an O/A ratio of 2/1. The circuit results are shown in Table 7, indicating reasonable extraction and excellent stripping efficiency.

Table 7 - Continuous mixer steeler tests using Alamine 336
Water dilution 2:1 (water:eluate) Feed 2.3 g U/L

Extraction circuit				Strip circuit								
						Strip solution						
O/A ratio	Raffinate U (g/L)	Organic u (g/L)	% Ext'n U	O/A ratio	Strip solvent U (g/L)	U	Th	Fe	Si	Ca	Mg	% Stripped U
1/1	0.16	2.23	93.5	4/1	0.0065	9.46	0.001	0.0008	0.004	0.0001	0.042	99.7

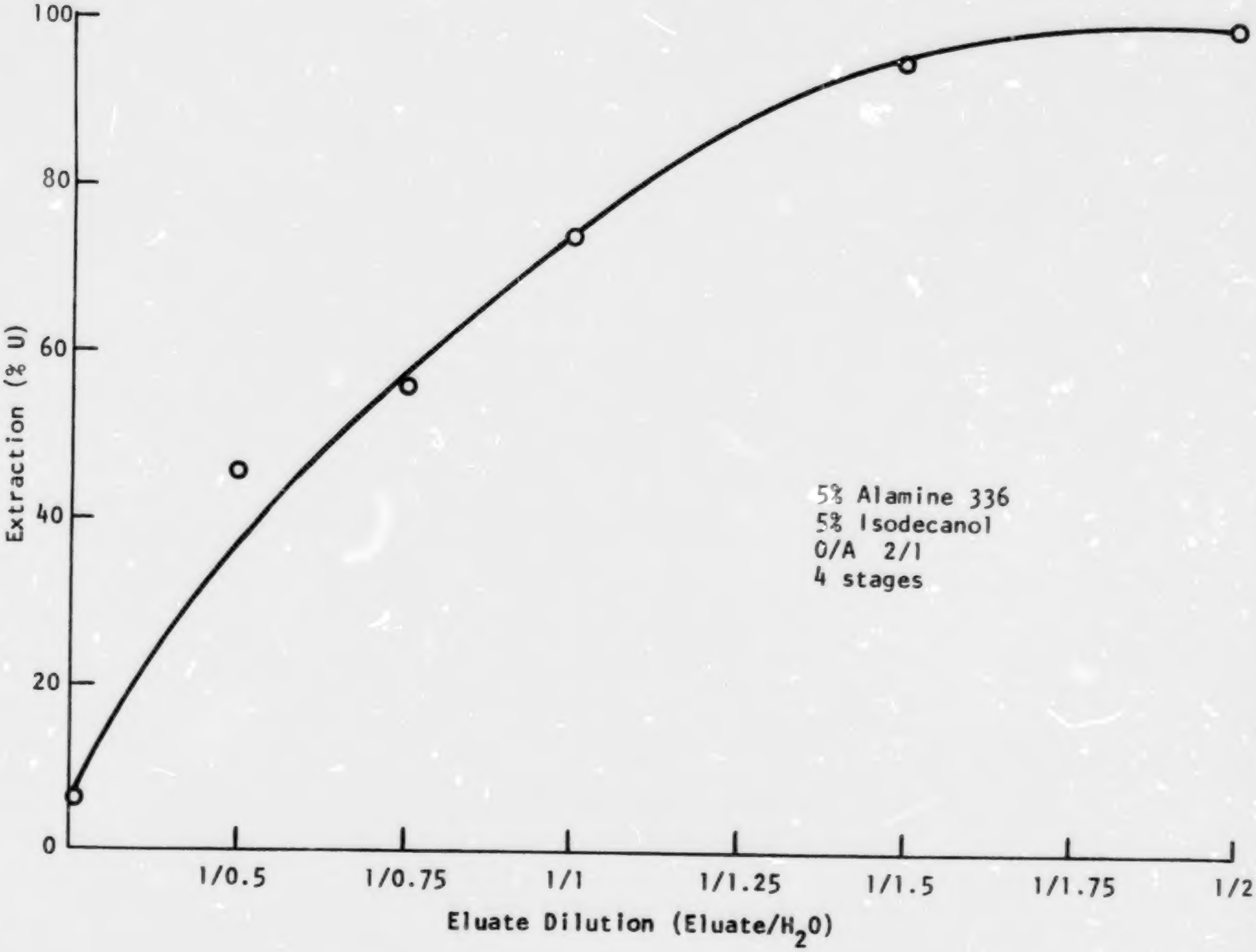

Figure 1. Effect of Eluate Dilution on U Extraction with Amine

Finally, the circuit was run for 3 days (extraction O/A 1.2/1) with no scrub stage, to determine whether any problems would develop over an extended running period. The mixing time was 1.5 minutes. The circuit performed well and no crud was generated. Comparison of the recycled solvent after 110 cycles with fresh solvent showed no difference in loading capacity; thus there was no apparent degradation. Solvent losses in solution were analyzed at 2.3 ppm amine.

DEHPA CIRCUIT

In addition to tests on the diluted feed in contact with the tertiary amine, continuous mixer settler tests were carried out on the alkylphosphate system. The feed solution at 7.24 g U/L was contacted with a solvent consisting of 3.3% DEHPA + 5% TBP in Shell 140 kerosene, which was previously equilibrated with 5% H_2SO_4 in two stages. Four stages of extraction were followed by four stages of stripping with 5% Na_2CO_3 solution. No scrub stage was used. The extraction and stripping results are shown in Table 8, indicating excellent uranium extraction and stripping.

Table 8 - Continuous mixer settler tests using DEHPA
Feed 7.24 g U/L

Extraction circuit				Strip circuit					
						Strip solution			
O/A ratio	Raffinate U (g/L)	Organic u (g/L)	% Ext'n U	O/A ratio	Strip solvent U (g/L)	U	Th	Fe	Si
1.5/1	0.0014	4.77	99.98	2/1	0.10	10.2	0.13	0.002	0.007

FINAL PRODUCT

The strip solution from the amine and alkylphosphate circuits were subjected to precipitation of the uranium "yellowcake" by the procedure previously described. The dried final products were analyzed, and a comparison of the relative product purity, together with the analysis of the regular Madawaska yellowcake product, are shown in Table 9. With respect to the yellowcake products from the amine extraction tests, it appears that water scrubbing such a loaded solvent from an eluex circuit is not necessary (as is often not the case if solvent extraction treatment of the leach liquor is directly applied). The products were all higher in purity than the Madawaska eluate precipitated without solvent extraction processing. The use of DEHPA as the extractant resulted in the highest grade uranium product, although containing slightly higher quantities of Fe, Si and Th.

Table 9 - Yellowcake analyses (%)

SX reagent	Product	Total Fe	SiO_2	Mg	Mo	Th	P_2O_5	U
No SX	Madawaska product	1.51	1.34	2.4	<0.01	1.07	0.47	68.1
Alamine	Without H_2O scrub	0.03	<0.01	1.58	<0.01	0.004	0.17	69.6
Alamine	With H_2O scrub	0.02	<0.01	1.36	<0.01	0.004	0.22	70.2
Alamine	Final run (no scrub), 3 days	0.01	0.01	0.38	<0.01	0.002	0.0016	70.8
DEHPA	No scrub stage	0.32	0.16	0.05	<0.01	0.01	0.003	71.3

All yellowcake products responded equally well in passing the amenability tests, with no gel formation. The break times for the amenability tests are given in Table 10 for three successive contacts in separatory funnels, 3 minutes, A/O 1/1.

Table 10 - Amenability test break times

	Contact Number		
	1	2	3
Initial break time	3 sec	3 sec	3 sec
Top interference break time	3	2	2
Initial interface formation time	12	10	10
Final interface formation time	25	25	25

DISCUSSION

Ion exchange eluates produced at Madawaska Mines, and possibly other uranium mills, that have a comparatively high silica content can be successfully treated by adapting an eluex process. Even with the relatively high concentration of chloride in the feed solution, an eluex process can be used. There were four eluex processes that appeared feasible and would seem to offer some potential as they all produced yellowcake products that formed no gel when evaluated by the amenability procedure of the Eldorado Nuclear Refinery. Each of the four extraction systems had something to offer and because the mechanism of extraction was somewhat different for each, different products would result because of differences in selectivity. Also, there were differences in kinetics, equilibrium and stages, ease of stripping, and synergism.

The synergistic system of Alamine 336 + TOPO, although producing excellent extractability of uranium, was nevertheless not the best system for stripping. Sodium oxalate could be used, followed by peroxide precipitation, but the product grade was low. Thus that system could not be considered until a more suitable stripping procedure is developed.

The present eluate contains almost 1 M NaCl, and if the chloride ion is increased, by the addition of more NaCl, then a solvating extraction of uranium can be achieved, followed by easy stripping with water. However, the system, at the conditions investigated, required 75% TBP to extract 83% of the uranium in one stage. Thus, in order to seriously consider that extractant, the reagent concentration should be lowered (to reduce solvent losses) and also the efficiency must be improved (stage requirements reduced). Further work would be required on the system.

The use of DEHPA appears to offer the potential of fast rates of extraction and because it is a cation extractant, chloride and sulphate will be rejected. However, because of the cation exchange mechanism, any cations such as thorium, iron, calcium, magnesium, rare earths, etc. that might be in the original eluate could be co-extracted. The one preliminary continuous run indicated no real physical problems, and the yellowcake was of acceptable quality. Because the extractant has many suitable qualities, it deserves further examination, perhaps with TOPO added as a modifier-synergist.

The dilution of the feed eluate solution (thereby diluting the chloride content) followed by the anion extraction with a tertiary amine such as Alamine 336 or Adogen 364 offers a process that could be immediately implemented. The continuous testing in the mixer settlers indicated no emulsion or degradation problems and produced a yellowcake product with much lower impurities compared to the mill product. The circuit would consist of 4-stages of extraction, 3-stages of stripping and 1-stage of acid equilibration.

CONCLUSION

The work has demonstrated the possibility of employing an eluex process for the treatment of the Madawaska eluate, producing a yellowcake of high purity that passes the Eldorado amenability test. While the dilution of the feed followed by solvent extraction is the process most suitable at the present time, the DEHPA and TBP systems may be equally or more effective if subsequent work is successful. A cost evaluation comparison should eventually be made on the four eluex possibilities.

APPENDIX I

Initial Break Time

This is noted by bringing the separatory funnel upright and immediately looking for a phase break near the stopcock. If solids are present, this may be difficult to observe. Look for a breaking swirl as the organic phase tends to sweep solids up with it. If the initial break is missed, immediately begin shaking for about 30 more seconds and try again. If solids prevent observation of the break, simply note that the break was not observable.

Top Surface Break Time

This is the phase break at the top surface. It may be the same as the initial break time, but it usually is slower due to frothing at the surface. This may be useful if the initial break time was unobservable.

Initial Interphase Formation Time

This is the time when the upper and lower phase lines meet in the area of the final interphase. In some samples, bubbles and froth will settle near the final interphase, and some time will elapse before final separation occurs. In other samples, there will be few bubbles, and the upper and lower lines will come together and stabilize quickly. In the latter case, the time recorded for initial and final interphase formation will be the same.

Final Interphase Formation Time

This is the final separation time. This is recorded when the final interphase is stable, and does not rise or fall from its final position in the separation funnel. In some samples in which froth or bubbles are present in the organic phase, total elapsed time may be several minutes. All bubbles should be dissipated. Surface tension problems may arise in the presence of solids, which makes this time difficult to judge. No shaking.

Summary

The elapsed time from the initial break time to the final interphase formation is the total elapsed time.

After observing the initial break (against a white or light background), the stopper on the separatory funnel should be removed, and the funnel placed in a clamped holder. All further observations are made against a white background.

ASSESSMENT OF
PRECONCENTRATION OF URANIUM ORES

PAPER PREPARED FOR PRESENTATION AT
CANADIAN URANIUM PRODUCERS METALLURGICAL COMMITTEE
1981 ANNUAL MEETING
MAY 19, 1981

SASKATOON, SASKATCHEWAN

by

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MINERALS DIVISION

1.0 INTRODUCTION

Because of the high cost of leaching, preconcentration of uranium ores prior to leaching has intrigued uranium metallurgists for many years. Preconcentration by flotation, heavy medium and sorting techniques have been investigated extensively in Canada. However, to the best of our knowledge, sorting is the only method which has been applied commercially in Canada.

C.M. Lapointe invented a radiometric sorter in the 1940's known as the "Lapointe Picker" and this was used by Eldorado Mines Ltd. at Port Radium and Port Hope. Sorting technology was advanced by L. Kelly and J. Hutter, who developed the K&H electronic sorter at Bicroft Uranium Mines at Bancroft, Ontario. The first commercial unit was put into commercial service at Bicroft in 1958. Subsequently, K&H sorters have been successfully employed on uranium ores in South Africa and Australia.

In recent years extensive preconcentration test work has been carried out at the CANMET laboratories of Energy, Mines and Resources Canada in Ottawa. The objective of this program was to develop beneficiation processes which would yield high recoveries of uranium, pyrite, radium and thorium in concentrates and produce chemically stable tailings with low levels of radioactivity.

Lummus Canada Inc. was commissioned by CANMET late in 1980 to evaluate preconcentration of low grade uranium ore, based on a typical ore from Elliot Lake. A hydrochloric acid leaching process investigated earlier by Lummus was selected as a Base Case for comparison with two proposed preconcentration - hydrochloric acid leaching processes.

The leaching processes have not been investigated fully in the laboratory. Cost data developed during Lummus Canada's studies are in the order of magnitude range, intended primarily to assist CANMET in the allocation of funds for further process development.

2.0 PROCESS DESCRIPTIONS

Schematic flowsheets of the processes selected for study are shown on Fig. 1-3 inclusive.

In the Base Case the ore is ground and pyrite recovered by flotation. Pyrite is roasted and the roaster gas converted to sulphuric acid. Flotation tailings are combined with the roaster calcine and leached with dilute hydrochloric acid in two stages.

The leach slurry is dewatered and the uranium in the pregnant liquor is recovered by solvent extraction. Uranium is precipitated with magnesia and recovered as yellowcake for marketing.

Separation of thorium from the uranium barren solution is achieved in a second solvent extraction circuit. Precipitated as the oxalate and dried, thorium is packaged and stored for possible future sale. Following thorium extraction the waste liquor is treated to remove radium, which is collected in disposable containers for permanent storage. The radium circuit is still in the development stage and cost estimates for this circuit are tentative.

It is necessary to remove residual chloride and hydrochloric acid from the final barren liquor. This is accomplished in a circuit which also regenerates hydrochloric acid for use in leaching. This circuit consists essentially of four stages, namely:

1. In the first stage the barren is evaporated to about half its volume and the steam distilled. The distillate, containing a low concentration of acid is utilized as filter wash following leaching.

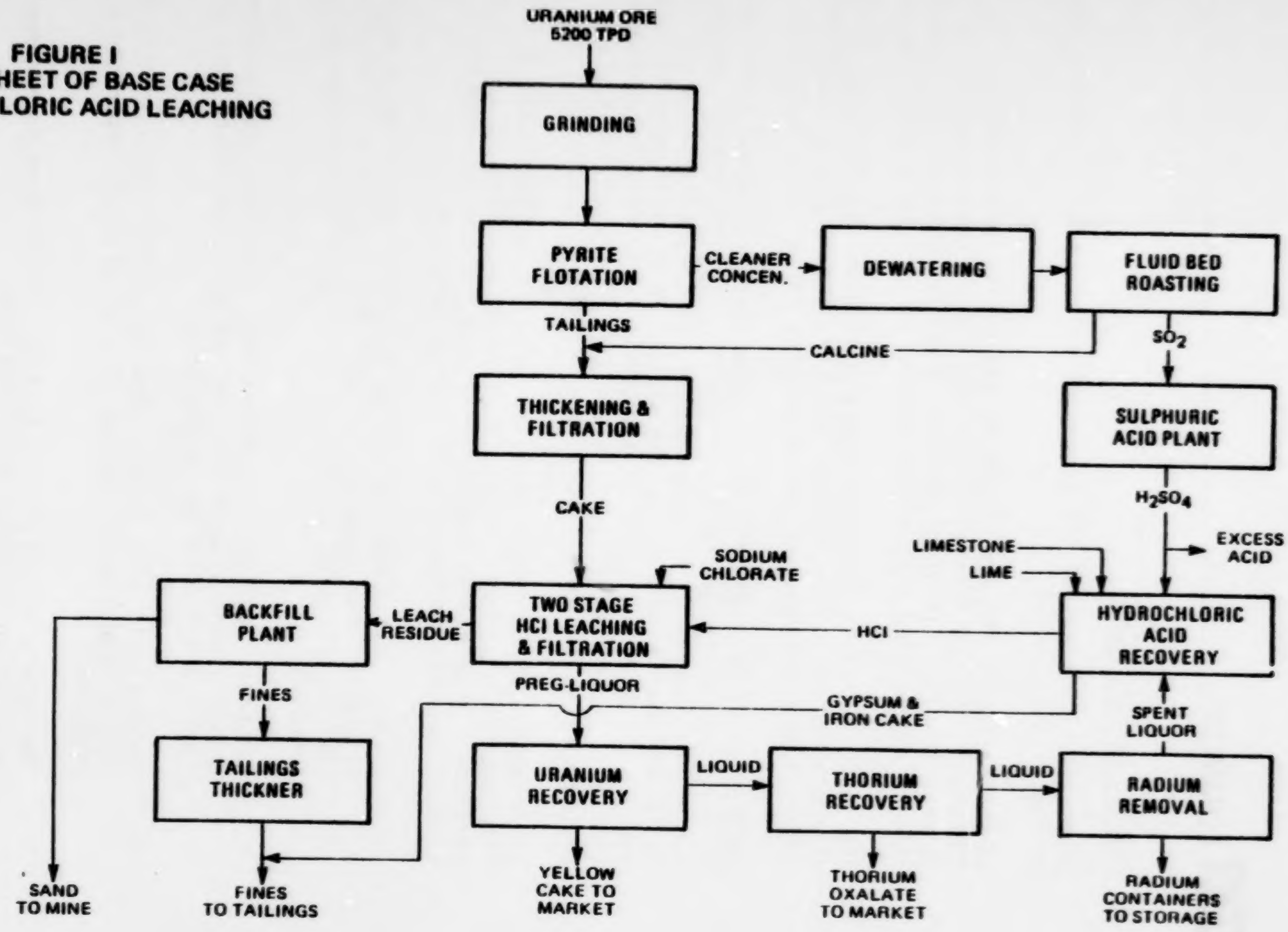
2. Evaporator bottoms are neutralized with lime and limestone. Metallic constituents are precipitated, separated from the liquor and discharged to tailings.

3. Hydrochloric acid is generated by adding sulphuric acid to the neutralized liquor. Gypsum is precipitated, separated and discarded.

4. The weakly acidic liquor is distilled to produce hydrochloric acid, which is recycled to leaching.

By cycloning repulped leach residues twice, sand product suitable for mine backfill is produced plus a slime product which is thickened and pumped to tailings. Up to 70% of the leach residue may be suitable for backfill or other purposes.

FIGURE I
FLOWSHEET OF BASE CASE
HYDROCHLORIC ACID LEACHING



2.0 PROCESS DESCRIPTIONS - Continued

In the utilities area two boilers produce high pressure steam, which is used to produce part of the power in the process. Low pressure steam from the turbines is used primarily for heating leach slurry and distilling hydrochloric acid. Additional steam is produced in a waste heat boiler on the roaster and is used to generate power and operate the blower in the acid plant.

For a more detailed description of the Base Case please refer to our paper entitled "Evaluation of Chlorine/Chloride Based Processes for Uranium Ores" which was presented at the same meeting as this paper.

The first preconcentration option selected for study is shown on Figure 2.

Grinding and pyrite flotation are identical to the Base Case except that pyrite cleaner tailings, instead of being recycled in flotation, are directed to leaching. Pyrite concentrate is roasted to produce sulphuric acid, as for the Base Case. Tailings from pyrite flotation flow directly to additional cells where a partial recovery of uranium is achieved by adding a small quantity of fatty acid.

Following classification at 150 mesh the uranium flotation tailings are treated by wet high intensity magnetic separation to recover additional uranium and other radioactive minerals (magnetic fraction).

The magnetic concentrates are combined with pyrite cleaner tailings, uranium flotation concentrate and roaster calcine, and the combined concentrates dewatered prior to leaching. (For cost purposes concentrate dewatering is included in the magnetic separation circuit).

Hydrochloric acid leaching and subsequent recovery circuits are functionally identical to those of the Base Case but circuits are smaller because of the reduced quantity of material leached.

Mine backfill requirements probably could be met by single stage cycloning of the coarse non-magnetic tailing product. However, to provide a fair comparison with the Base Case, it is assumed that the leach residue is also classified to produce backfill. Fine products are thickened and pumped to tailings.

Referring to Figure 3, the second preconcentration option is identical to the first except that uranium floation is not included in the flowsheeet.

FIGURE 2
FLOWSHEET OPTION NO.1
PRECONCENTRATION AND HYDROCHLORIC ACID LEACHING

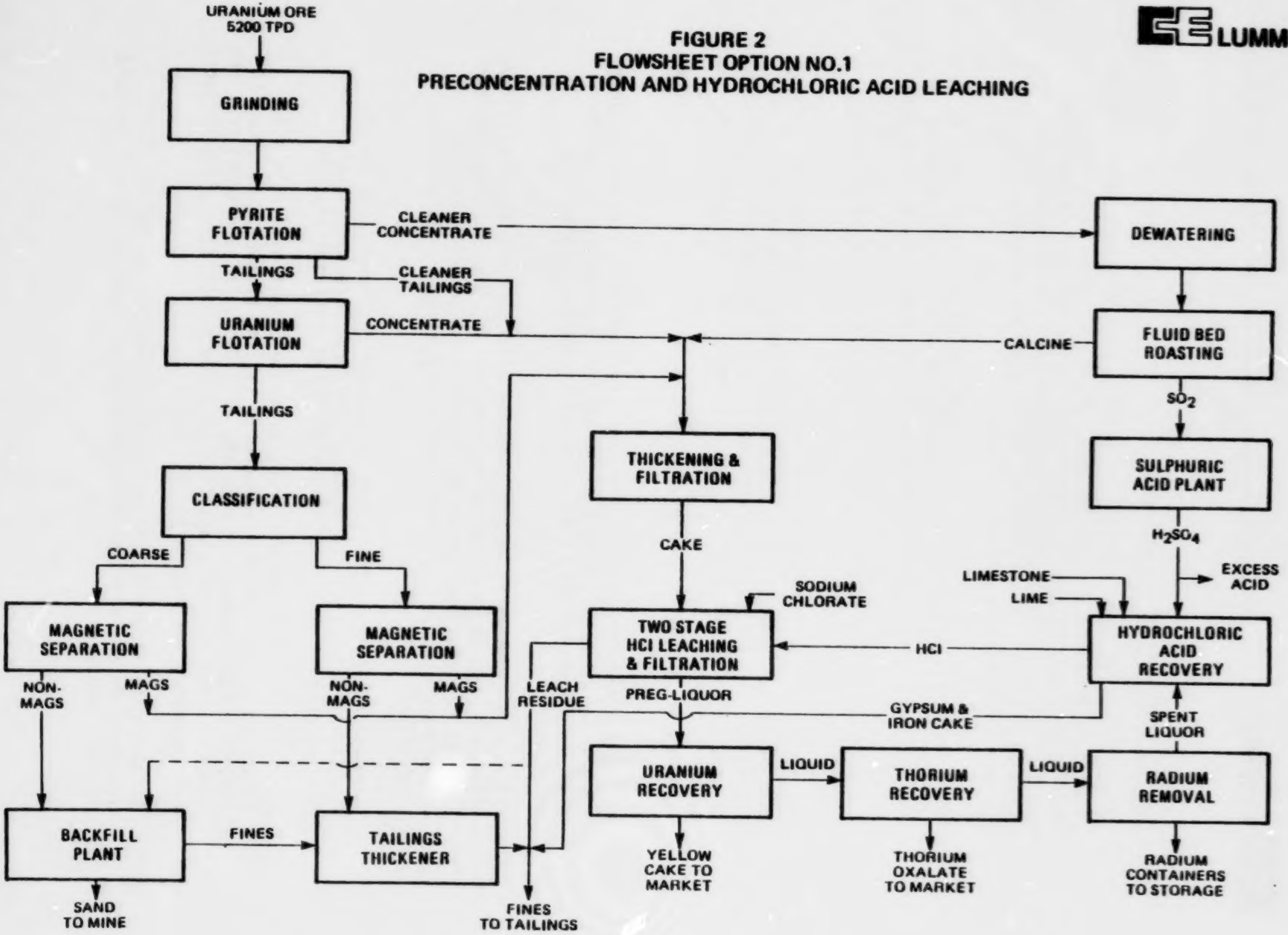
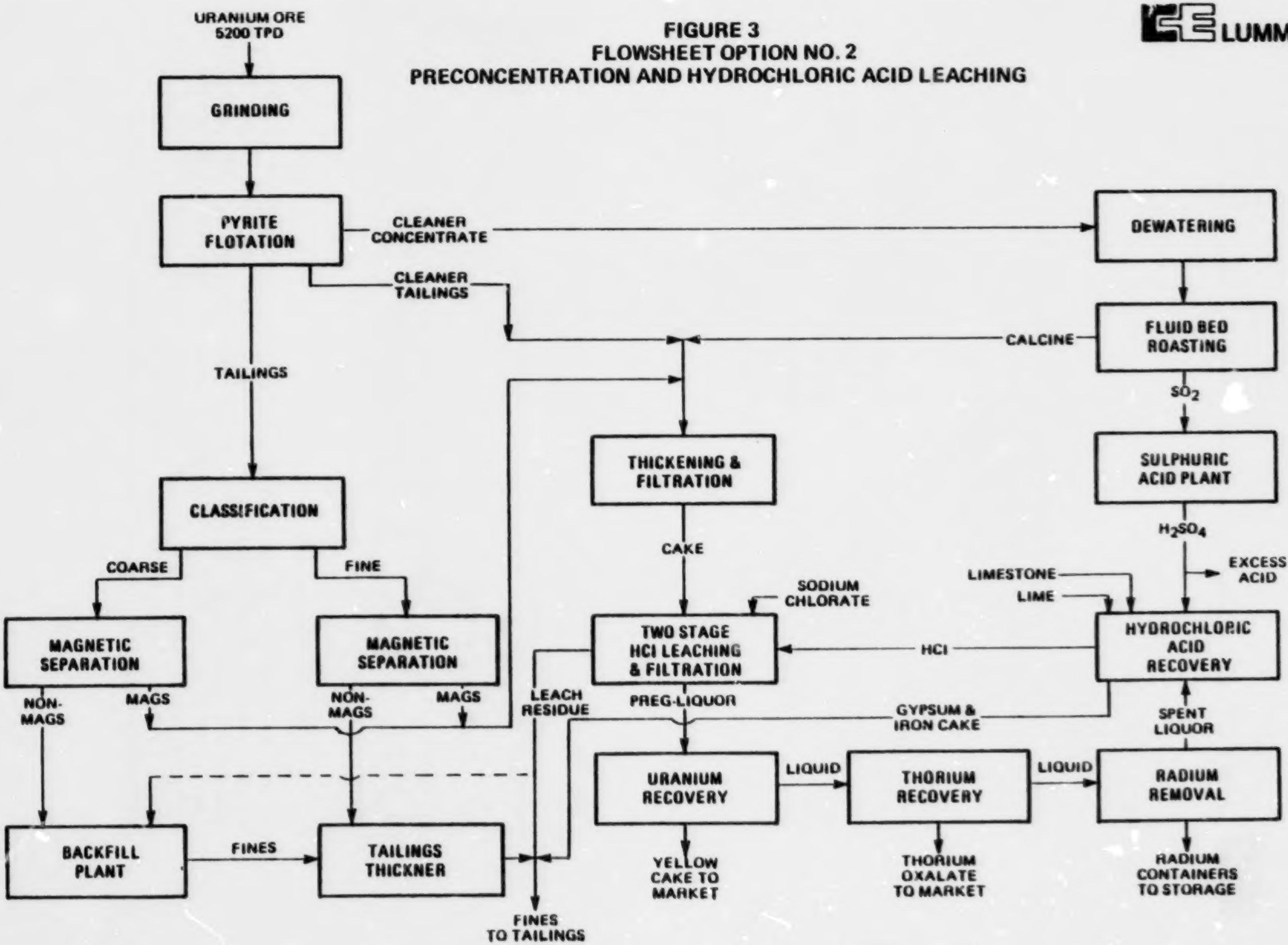


FIGURE 3
FLOWSHEET OPTION NO. 2
PRECONCENTRATION AND HYDROCHLORIC ACID LEACHING



3.0 PROCESS DESIGN

3.1 Metallurgy

Material balances for the preconcentration circuits are based on test results summarized in Table 1.

Mill feed in each case is as follows:

Milling rate	5200 Tons/Day
Design Operating Time	330 Days/Year
Mill Feed	1,716,000 Tons/Year
U_3O_8 in Mill Feed	0.10 %
	3,432,000 Lb/year

Calculated uranium recoveries are as follows:

	Base Case	Preconc'n Option No. 1	Preconc'n Option No. 2
Recovery % - Preconc.	-	96.5	95.3
- Leaching	98.0	98.0	98.0
- Overall	98.0	94.6	93.4
U_3O_8 Produced, Lb/year	3,363,000	3,246,000	3,205,000
Preconcentration Loss, Lb/year	-	117,000	158,000

3.2 Acid Consumption

Consumptions of hydrochloric acid (as 100%) in leaching were 66, 37 and 33 lb/ton of plant feed for the Base Case, Option No. 1 and Option No. 2 respectively. Limestone and lime consumptions in neutralization are directly proportional to acid consumptions.

3.3 Dewatering Equipment

It was evident from preconcentration data that concentrates contained a higher proportion of fines than the original ore and, therefore, settling and filtration design figures used for the ore might not be applicable to the concentrates. Accordingly, tests were carried out to determine comparative thickener and filter areas for neutral slurries and filter areas for acid leach slurries. Based on fairly limited test data the following relative areas were used for the dewatering circuits:

TABLE 1

PRECONCENTRATION METALLURGY

PRODUCT	WEIGHT %	ASSAY, % *				DISTRIBUTION%			
		U	Th	Ra	S	U	Th	Ra	S
Flowsheet No. 1									
Pyrite Cl. Conc.	6.3	0.12	0.074	390	38.65	9.6	13.5	9.1	88.9
Py.Cl. Tails + Mags + U Conc	38.2	0.18	0.069	610	0.72	86.9	75.4	86.4	10.0
+150 Non-Mags	37.0	0.0025	0.005	15	0.03	1.2	5.4	2.1	0.4
-150 Non-Mags	18.5	0.010	0.011	35	0.10	2.3	5.7	2.4	0.7
HEAD (CALC)	100.0	0.079**	0.035	270	2.73	100.0	100.0	100.0	100.0
Flowsheet No. 2									
Pyrite Cl. Conc.	6.3	0.12	0.074	390	38.65	9.6	12.8	9.3	89.9
Py.Cl. Tails + Mags	31.0	0.22	0.085	720	0.76	85.7	71.7	84.2	8.7
+150 Non-Mags	37.0	0.0025	0.005	15	0.03	1.2	5.2	2.1	0.4
-150 Non-Mags	25.7	0.011	0.015	45	0.11	3.5	10.3	4.4	1.0
HEAD (CALC)	100.0	0.079**	0.037	264	2.70	100.0	100.0	100.0	100.0

* Radium in picocuries

** 0.093% U_3O_8

3.0 PROCESS DESIGN

3.3 Dewatering Equipment - Continued

	Neutral Thickener Area	Neutral Filter Area	Acid Filter Area
Base Case	1.00	1.00	1.00
Preconc. Option No. 1	0.75	0.50	1.00
Preconc. Option No. 2	0.75	0.40	1.00

Filter cakes from concentrates contained higher moistures than from the total ore, as shown by the following comparison:

	Cake Moisture, %	
	Neutral Filter	Acid Filter
Base Case	13.4	18.9
Preconc. Option No. 1	20.8	26.1, 24.3
Preconc. Option No. 2	18.4	25.1, 22.2

4.0 CAPITAL COST COMPARISON

Capital costs were estimated for battery limit milling plants located in Central Canada. Tailings pumps are included but no allowance made for tailings lines or disposal area. Land costs and owner's costs during construction are not included. Costs are in current dollars.

A capital cost comparison of the two preconcentration options with the Base Case is shown in Table 2.

TABLE 2 - CAPITAL COST COMPARISON

	Costs in $ Millions		
	Leach Total Ore	Preconc. F/S No. 1	Preconc. F/S No. 2
Installed Cost of Equipment			
Grinding and flotation	15.9	9.9	9.4
Roasting and sulphuric acid plant	23.6	23.6	23.6
Magnetic separation	-	10.6	12.5
Leaching with hydrochloric acid	39.7	36.9	36.3
Separation and precipitation of uranium as yellowcake	6.7	6.4	6.4
Separation and purification of thorium and radium	9.4	9.0	9.0
Regeneration of hydrochloric acid	12.7	9.2	8.5
Mine backfill	2.9	2.8	2.8
Sub-Total: Installed Equipment	110.9	198.4	108.5
Cost of Buildings	18.2	18.2	18.2
Cost of Utilities	9.7	5.8	5.8
TOTAL DIRECT PLANT COST	138.8	132.4	132.5

5.0 OPERATING COST COMPARISON

Operating costs are in current dollars and exclude taxes, interest payments, depreciation, head office and marketing costs.

Operating costs for the Base Case and the two preconcentration flowsheets are summarized in Table 3.

6.0 DISCUSSION

Results in Sections 4 and 5 indicate that preconcentration results in appreciable savings in capital and operating costs although there is little to choose between the two preconcentration options. Operating cost savings are largely offset by uranium losses in preconcentration, as discussed later.

6.1 Capital Costs

Compared with the Base Case, overall capital cost savings of $6.4 million and $6.3 million are shown for preconcentration Options No. 1 and No. 2 respectively.

The apparent decreases in capital costs in grinding and flotation in Options No. 1 and No. 2 are not significant; they result from a transfer of neutral thickening equipment from grinding to the magnetic separation area.

Savings in the leaching area are fairly modest and are due largely to smaller leaching vessels. Filtration represents a large portion of the cost of this circuit and it is not affected by preconcentration.

Uranium, thorium and radium recovery areas are largely unaffected by preconcentration because, apart from the initial extraction phase in each case, costs are largely product-dependent.

Decreased costs in hydrochloric acid regeneration amount to $3.5 million and $4.2 million for Options No. 1 and No. 2 respectively. These savings result from lower solution flow rates and smaller equipment.

In the utilities area, capital cost savings of approximately $4 million in the preconcentration processes stem from decreased steam consumption in leaching and hydrochloric acid regeneration.

TABLE 3

OPERATING COST COMPARISON

		ANNUAL CONSUMPTION			ANNUAL COST $ MILLIONS		
ITEM	UNITS	BASE CASE	F/S NO. 1	F/S NO. 2	BASE CASE	F/S NO. 1	F/S NO. 2
Wages and Salaries	No.	(310)	(315)	(315)	7.30	7.41	7.41
Electrical Energy	KWH x 10^6	54.6	69.7	72.9	1.46	1.86	1.95
Steam	Lb x 10^6	1,618	1,025	953	4.87	3.08	2.86
Process Water	Gal x 10^6	240	240	240	0.24	0.24	0.24
Maintenance Materials	-	-	-	-	5.64	5.47	5.47
Major Chemicals	-	-	-	-	4.42	2.11	1.86
Oper. Reagents and Supplies	-	-	-	-	1.24	1.32	1.11
TOTAL DIRECT OPERATING COST					25.17	21.49	20.90
Cost per Ton of Ore					14.67	12.52	12.18
Cost per Pound of U_3O_8					7.48	6.62	6.52

6.0 DISCUSSION - Continued

6.2 Operating Costs

Preconcentration has little affect on operating costs apart from electrical energy, steam and major chemicals.

Overall power consumption is not affected significantly by preconcentration, however, since less power is generated from steam, the cost of purchased power increases.

Reduced flowrates in leaching and hydrochloric acid regeneration result in substantial savings in steam cost, amounting to $1.8 million and $2.0 million for Options No. 1 and No. 2 respectively. At least half of these savings are in hydrochloric acid regeneration.

Considering major chemicals, an indicated annual saving in sodium chlorate of approximately one million dollars in each preconcentration option is based on a laboratory reagent addition of 2.5 kg/tonne of leach feed. This rate is high compared with current practice in sulphuric acid leaching plants. If further test shows that the rate can be lowered, indicated savings in sodium chlorate in the preconcentration options will be reduced.

Savings in lime and limestone are substantial. These are directly proportional to reductions in consumptions of hydrochloric acid.

6.3 Uranium Losses in Preconcentration

Uranium losses in preconcentration amount to 117,000 lb. U_3O_8/year and 158,000 lb. U_3O_8/year for Options No. 1 and No. 2 respectively. Assuming a value of $30/lb, uranium losses are valued at $3.5 million and $4.7 million for Options No. 1 and No. 2 respectively.

6.0 DISCUSSION - Continued

6.4 Overall Evaluation

Cost advantages and disadvantages of preconcentration are summarized below. All costs are in millions of dollars.

	Base Case	Preconc'n Option No. 1	Preconc'n Option No. 2
Capital Costs			
Total Direct Plant Cost	138.8	132.4	132.5
Saving by Preconcentration	-	6.4	6.3
Annual Operating Costs			
Direct Operating Cost	25.2	21.5	20.9
Saving by Preconcentration	-	3.7	4.3
Annual Uranium Losses			
Preconcentration Losses, lb U_3O_8	-	117,000	158,000
Value at $30/lb		3.5	4.7

Savings in operating costs from preconcentration are offset, almost equally, by the value of uranium losses. However, a capital saving of approximately six million dollars is indicated for each preconcentration option.

6.5 Preconcentration and Sulphuric Acid Leaching

Preconcentration in a conventional slulphuric acid leaching plant was not included in the scope of the study, however the cost figures developed in the study indicate that there would probably be no capital cost advantage due to preconcentration. Considering operating costs and uranium losses to-gether, savings due to preconcentration could amount to one million dollars. This assumes that sulphuric acid for leaching is purchased at a cost of $50/ton.

Accordingly preconcentrration could be marginally attractive in a sulphuric acid leaching plant.

The preconcentration tailing, having a low content of radioactive elements, appears to be well suited as a source of mine backfill.

7.0 RECOMMENDATIONS

It was recommended that test work on preconcentration be continued, particularly to provide more definitive process design data in some areas.

Although it has been demonstrated that high intensity magnetic separation is a very effective method of recovering radioactive minerals, the machines are quite expensive. Preliminary estimates for the machines alone are $4.0 million and $5.0 million for Options No. 1 and No. 2 respectively. Further investigations, preferably with a continuous pilot unit, were recommended to determine whether the cost of the machines could be reduced.

8.0 ACKNOWLEDGEMENTS

The author wishes to thank the management of Lummus Canada Inc. for permission to publish this paper. We are grateful to CANMET engineers and scientists, particularly W.A. Gow, G.M. Ritcey, Dr. K.E. Haque, D. Raecevic and M. Raecevic for their co-operation and assistance with the study. We also acknowledge the important role played in the study by Dr. G. Angelov, who was retained by Lummus to assist with the study.

DISCUSSION OF PAPER BY I. EDWARDS (PRECONCENTRATION)

A. Moorish	Preconcentration appears to be a fine concept for Saskatchewan and could become important in the next 20 years.
M. Campbell	Should not charge $30/lb U losses.
A. Oliver	What is the cause for the decrease in utility costs?
I. Edwards	Steam consumption is the major cost.

ASSESSMENT
OF CHLORINE/CHLORIDE PROCESSES
FOR URANIUM ORES

Paper Prepared for Presentation at
Canadian Uranium Producers Metallurgical Committee
1981 Annual Meeting
May 19, 1981
Saskatoon, Saskatchewan

by

I.C. Edwards

CE LUMMUS
MINERALS DIVISION

1.0 INTRODUCTION

Conventional acid leaching processes for uranium ores, coupled with waste management procedures, result in essentially all the radium and thorium in the ores ending up in the solid tailings waste. Uranium which is not dissolved during leaching also reports in the solid wastes. Concern has been expressed that, in the long term, these radioactive elements may leach out of tailings deposits in concentrations harmful to the environment.

There is also concern about acid seepage from tailings. Most Canadian uranium ores contain some sulphide minerals, commonly pyrite. In the presence of oxygen and bacteria, pyrite gradually oxidizes to form sulphates of iron and sulphuric acid. These oxidation products promote the dissolution of heavy metal components and radionuclides in the tailings.

Various proposals have been made to deal with the problem of radioactive elements in uranium tailings wastes. The CANMET laboratories of the Department of Energy, Mines and Resources Canada, in Ottawa have been investigating alternative processing methods for uranium ores in which the hazardous radioactive elements are extracted during processing to yield tailings wastes with low levels of radioactivity. The radioactive elements are concentrated in forms suitable for long term storage in areas designed to accommodate radioactive wastes, e.g. in stable underground caverns. Waste storage is outside the scope of this paper.

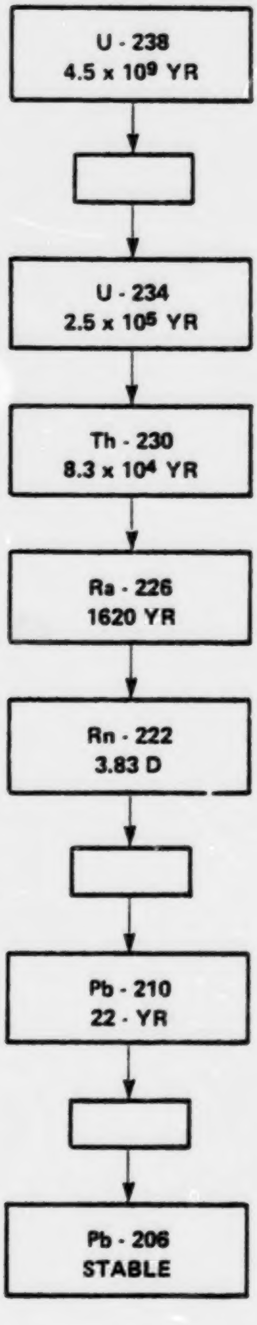
The radionuclides in uranium ores consist essentially of daughter elements of uranium-238. A simplified diagram showing the radioactive decay process for U-238 is shown on Figure 1. Each small block in the chart represents one or more radionuclides with very short half-lives and, therefore, present in very small concentrations. The only elements with long half-lives are uranium-238, uranium-234, thorium-230 and radium-226. By extracting these elements from the ore, long-term radiation from the tailings is effectively terminated. If Pb-210 is not extracted it will disintegrate fairly quickly into stable lead-206 because it has a half-life of only 22 years.

Various methods can be used to control acid formation in tailings ponds. The most effective method is to remove pyrite by flotation, preferably prior to leaching, and roast it to produce sulphuric acid.

Lummus Canada Inc. was commissioned by CANMET to evaluate two uranium extraction processes being developed by CANMET to overcome the alleged problems with tailings from existing acid leaching processes.

FIGURE I

RADIOACTIVE DECAY - URANIUM SERIES



SMALL BLOCKS REPRESENT ELEMENTS WITH HALF-LIVES OF < 1 SEC. TO 138 D.

1.0 INTRODUCTION - Continued

One process, in which the ore is leached in aqueous hydrochloric acid, may be applied to low grade ores such as those occuring in the Elliot Lake and Bancroft areas of Ontario.

The second process, involving gaseous chlorination of the ore, is an expensive process which may be suitable for high grade ores, such as those found in Saskatchewan.

The purpose of the Lummus study was primarily to assist CANMET in the allocation of funds for further process development. Neither of the processes has been studied fully in the laboratory. Cost data developed during the study are, therefore, in the order of magnitude range.

2.0 STUDY CRITERIA

The following criteria were used for selection of the processes for evaluation:

1. Radium to be extracted in a form suitable for long term storage.

   Tailings radioactivity objective:

   Low Grade Ore: 15 picocuries per gram (pCi/g)
   High Grade Ore: as low as practically possible.

2. Thorium extraction to be as high as practically possible.

3. Uranium extraction at least as high as conventional acid leaching processes which extract about 95% of the uranium.

4. Minimum effect on the environment from elements in the ore or introduced during the process.

For study purposes the following ore analyses were assumed:

Element	Analysis, % Low Grade	Analysis, % High Grade
U_3O_8	0.1	2.5
Th	0.044	0.01-0.02
Ra	350 pCi/g	6,000 pCi/g
Fe	3.5	3.6
Si	39	27
S	3.0	1.2
As	--	4.0
Ni	--	2.0

The low grade analysis is typical of Elliot Lake ores. The high grade analysis is more or less typical of Saskatchewan ores; however, these show wide variations in mineral composition and uranium grade.

In view of the low thorium content in the Saskatchewan ores tested at CANMET thorium extraction by chlorination of these ores was not investigated.

Uranium production in each case is 10,000 lb/day of U_3O_8.

3.0 PROCESS DESCRIPTIONS

3.1 Hydrochloric Acid Leaching of Low Grade Ore

A general process flow diagram for the hydrochloric acid leaching process is shown in Figure 2.

Grinding and Pyrite Flotation

To achieve the required extraction efficiency, the ore is wet ground to 60% -200 mesh.

In order to minimize environmental problems, pyrite is floated with an efficiency of 90% or better, and the second cleaner flotation concentrate is roasted. Tailings are thickened and filtered prior to leaching.

Roasting of Pyrite and Production of Sulphuric Acid

Pyrite is roasted in a fluid-bed roaster, and the sulphur dioxide gas is converted to sulphuric acid. Acid production amounts to 142,000 tons/year, most of which is used to regenerate hydrochloric acid. It is assumed that the remainder, amounting to 33,000 tons/year is marketed. For study purposes the sulphuric acid is assumed to have no commercial value.

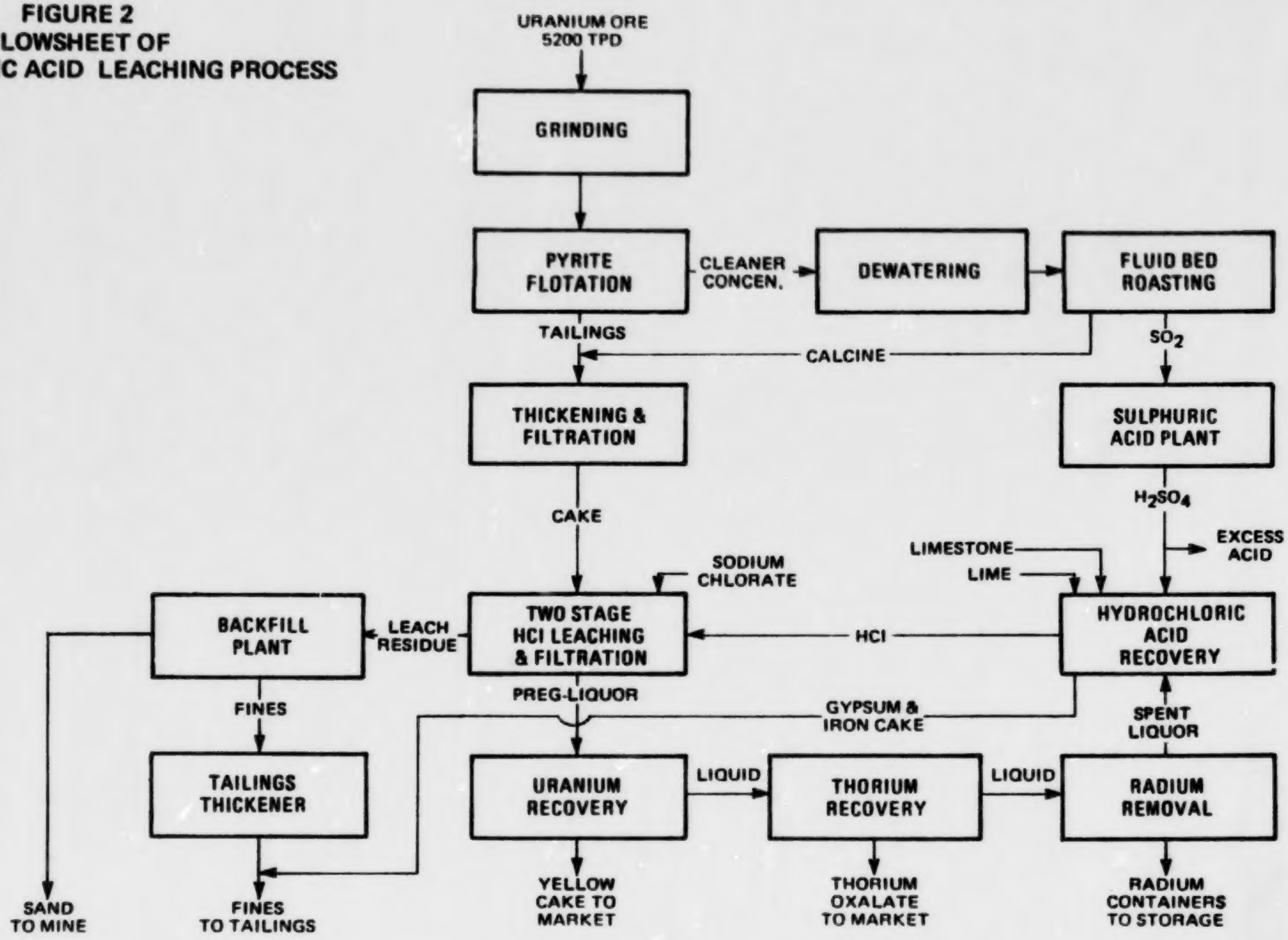
Hydrochloric Acid Leaching

Pyrite flotation tailings, together with roaster calcine, is leached in two stages, using hydrochloric acid. This dissolves uranium, thorium and radium. The leach slurry is filtered and washed to recover the radioactive minerals. A portion of the leach liquor is recycled to the head of the leaching circuit to minimize the quantity of liquor in subsequent treatment stages and the remainder advances to the uranium recovery circuit.

Separation and Precipitation of Uranium as Yellowcake

The pregnant liquor is clarified and treated in a solvent extraction circuit. Uranium is extracted in an organic phase containing tri-butyl phosphate (TBP) and kerosene. Uranium is stripped from the organic phase with water and is precipitated with a slurry of magnesia (MgO) in water. The slurry is filtered, washed and dried to meet yellowcake product specifications. Barren uranium solution (raffinate) from solvent extraction advances to the thorium circuit.

FIGURE 2
FLOWSHEET OF
HYDROCHLORIC ACID LEACHING PROCESS



3.0 PROCESS DESCRIPTIONS

3.1 Hydrochloric Acid Leaching of Low Grade Ore - Continued

Separation and Purification of Thorium and Radium

Separation of thorium from the uranium barren solution is achieved by solvent extraction using kerosene, alcohol, and an amine. Thorium is stripped with hydrochloric acid solution.

After stripping, the strip solution is evaporated in a single effect evaporator to remove excess hydrochloric acid and to concentrate thorium in solution. Thorium is precipitated as thorium oxalate by addition of oxalic acid solution. The precipitate is filtered and washed and the filter cake dried. The dry product is packed in suitable containers for storage for possible future marketing when thorium power reactors are developed.

At the time the study was done it was believed that a two-stage ion exchange process could be used to extract radium. It was proposed to collect radium in disposable columns which would be transported to a permanent storage facility. The viability of this process is now in doubt and cost figures for radium removal are, therefore, only tentative.

The barren liquor from the radium section, now depleted of radioactive elements is treated to recover hydrochloric acid.

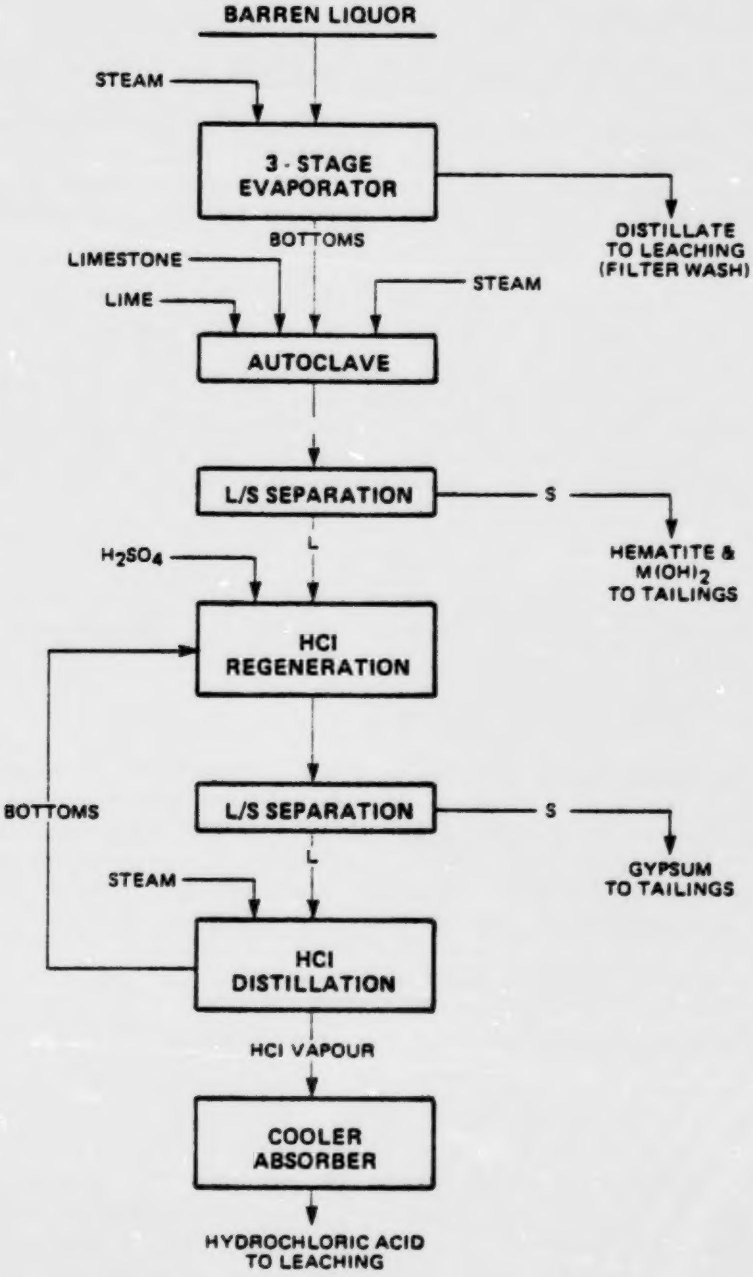
Regeneration of Hydrochloric Acid

The barren liquor contains metallic chlorides and free hydrochloric acid in concentrations far in excess of tolerable levels for direct discharge to the environment. In the regeneration circuit the metallic constituents are removed from the liquor and the chloride converted to hydrochloric acid for re-use in the leaching circuit. This is accomplished in several stages as shown on Figure 3.

In the first stage, about half the barren liquor is evaporated in a 3-stage evaporator in order to reduce the volume of liquor requiring further treatment. The distillate, containing a low concentration of hydrochloric acid, is utilized as filter wash water in the leaching circuit.

Evaporator bottoms are heated and treated in an autoclave with limestone and lime which neutralize the free acid and precipitate the metallic constituents as hydroxides. At the same time iron present is oxidized to hematite (Fe_2O_3) by injection of oxygen.

FIGURE 3
FLOWSHEET OF BARREN NEUTRALIZATION AND HYDROCHLORIC ACID REGENERATION



3.0 PROCESS DESCRIPTIONS

3.1 Hydrochloric Acid Leaching of Low Grade Ore - Continued

Slurry discharging from the autoclave, now containing metal hydroxides, hematite and calcium chloride is cooled and centrifuged to remove the metallic constituents, which are pumped to tailings. This liquid/solids separation is improved by the prior treatment to form hematite.

In the next stage, the centrate (liquor) from the centrifuges is treated with sulphuric acid in a series of regeneration tanks to convert the calcium chloride in solution into free hydrochloric acid and gypsum. The gypsum is removed by centrifuging and discarded to tailings. The liquor goes to a distillation tower where hydrochloric acid (32%) is distilled off and recycled to the leaching circuit. Bottoms from the column are recycled to the regeneration tanks.

Mine Backfill

Leach residue is classified by two-stage cycloning to produce a sand product suitable for mine backfill, and a fine product which is disposed of in a tailings pond.

Utilities

Two high pressure boilers produce steam. The high pressure steam is used to generate part of the power used in the process. Low pressure steam from the turbines is used primarily for heating leach slurry and distilling hydrochloric acid. Additional steam is produced in a waste heat boiler on the roaster. This steam is used to generate power and operate the blower in the acid plant.

3.2 Chlorination Process for High Grade Ore

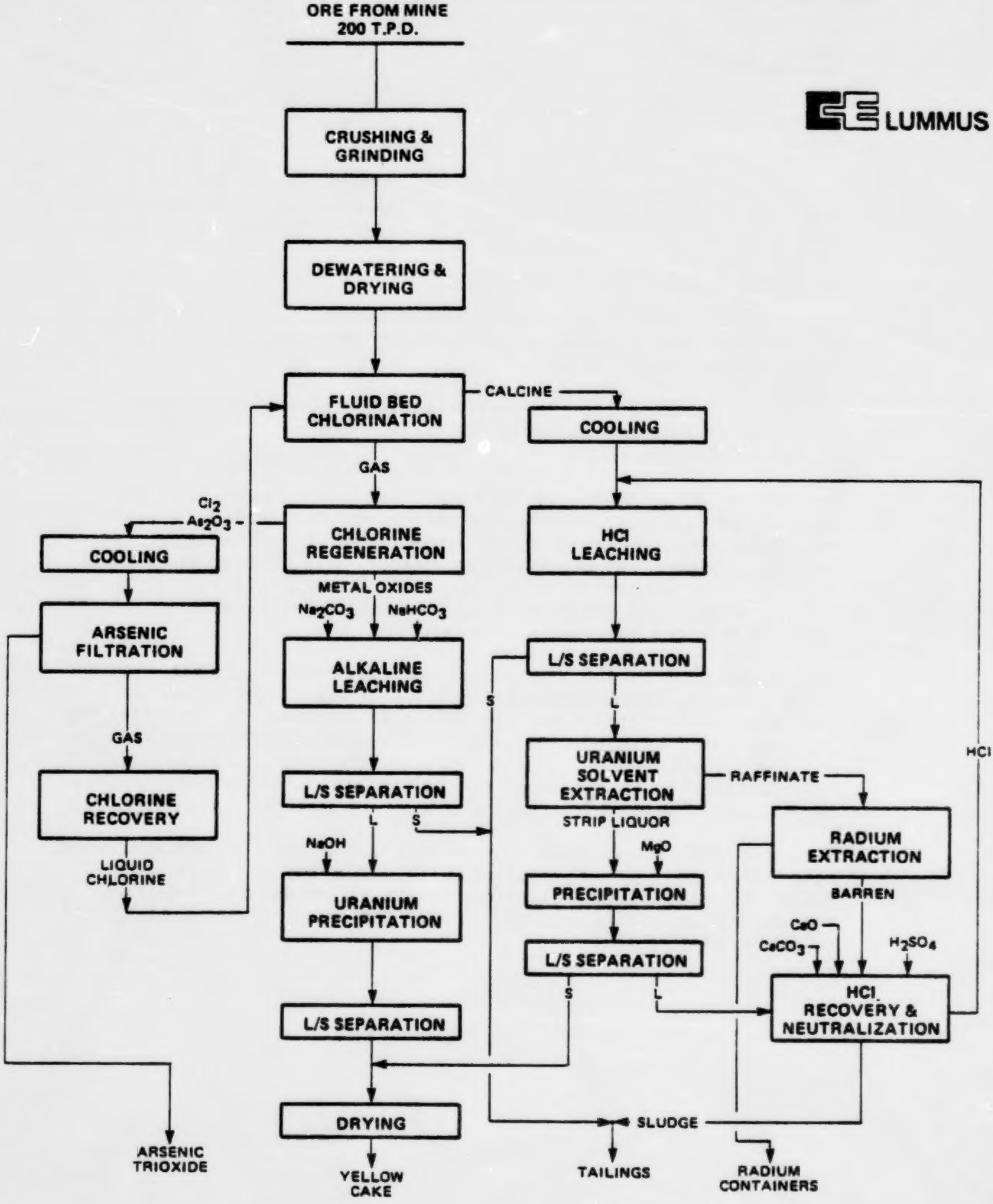
A general process flow diagram for the high grade ore is shown on Figure 4.

Grinding

To achieve satisfactory extraction of uranium, the ore is wet ground to 60% minus 200 mesh. Wet grinding is employed to minimize emissions of radioactive dust and radon gas. Ground slurry is thickened and filtered and the filter cake conveyed to a fluid bed dryer.

FIGURE 4
FLOWSHEET OF CHLORINATION PROCESS FOR HIGH GRADE URANIUM ORE

LUMMUS



3.0 PROCESS DESCRIPTIONS

3.2 Chlorination Process for High Grade Ore - Continued

Drying

In order to obtain high chlorination efficiency the ore has to be bone dry. The dryer bed product is conveyed to the chlorination section.

Chlorination and Chlorine Regeneration

Chlorination of the total ore is carried out in a series of two fluid bed reactors and a fluid bed cooler. The first reactor operates at 250°C, the second at 900°C and the cooler at 100°C.

Fluidization in the first two chlorination reactors is achieved with a mixture of nitrogen and chlorine. The nitrogen is a by-product of the on-site oxygen plant.

Exit gases from both chlorination reactors contain chlorides of uranium, iron, nickel and arsenic but no radium. (Radium chloride is not volatilized). These gases are passed to a chlorine regenerating reactor, in which oxygen reacts with the volatile metal chlorides and releases chlorine gas. All the metals are converted to oxides, which, with the exception of arsenic oxide, are solids at the high temperatures employed in the reactors.

The off-gases from the regenerator are passed through hot cyclones and filters to remove the solid oxides. These oxides, combined with the regenerator underflow product are conveyed to an alkaline leaching circuit to recover uranium. Arsenic oxide, which is still in the gaseous form, passes through the hot dust collectors. Subsequently, it is condensed by cooling and is collected in a cold bag filter. The arsenic trioxide product is packaged for sale.

The cleaned off-gas from the arsenic collectors is scrubbed with cold water to recover chlorine. The chlorine scrubber water is heated to distill the chlorine, which is dried and liquified by compression and refrigeration. Liquid chlorine is recycled to the chlorination circuit.

3.0 PROCESS DESCRIPTIONS

3.2 Chlorination Process for High Grade Ore - Continued

Alkaline Leaching

About 2/3 of the uranium in the ore is volatilized by chlorine. It is extracted from the regeneration reactor solids by a conventional carbonate-sodium bicarbonate leaching process. The uranium is precipitated from the leach liquor with caustic soda and is recovered as yellowcake.

Hydrochloric Acid Leaching

Calcine product from the chlorination reactors is leached with hydrochloric acid to extract non-volatile chlorides of uranium and radium. This leaching circuit flowsheet is similar to that used for low grade ore but the tonnage treated is much smaller.

Uranium is extracted from the leach solution by solvent extraction and precipitated with magnesia. The slurry is filtered and the filter cake combined with the alkaline leach uranium product prior to drying.

Extraction and Separation of Radium

Radium is extracted from the solvent extraction raffinate by two stages of ion exchange. The circuit is similar to that described earlier for low grade ore.

Hydrochloric Acid Recovery

The system used for recovery of hydrochloric acid is similar to that used in the low grade plant, but is much smaller because of the reduced flow rates involved.

Tailings Disposal

Residues from both leaching circuits are slurried with water and pumped to a tailings pond.

Mine backfill is not produced.

4.0 EVALUATION OF HYDROCHLORIC ACID LEACHING PROCESS

4.1 Capital Costs

Capital costs were estimated for a battery limits leaching plant from receipt of ore from the mine to packaging of products. Tailings pumps are included but no allowance made for tailings lines or disposal area.

Owner's costs during design and construction are not included.

Costs are in current dollars and are based on an installation in a developed area of Central Canada.

Capital costs are estimated to be as follows:

Installed cost of equiment:	$ Millions
Grinding and flotation of pyrite	15.9
Roasting of pyrite and sulphuric acid plant	23.6
Leaching with hydrochloric acid	39.7
Separation and precipitation of uranium as yellowcake	6.7
Separation and purification of thorium and radium	9.4
Regeneration of hydrochloric acid	12.7
Mine backfill	2.9
Sub-Total: Installed Equipment	110.9
Cost of Buildings:	18.2
Cost of Utilities:	9.7
Total Installed Plant Cost	138.8

4.0 EVALUATION OF HYDROCHLORIC ACID LEACHING PROCESS - Continued

4.2 Operating Cost Estimates

Operating costs are in current dollars and exclude taxes, interest payments, depreciation, head office and marketing costs. Annual operating costs are estimated to be as follows:

Item	Units	Annual Consumption	Cost $ Millions
Wages & Salaries	No.	(310)	7.30
Maintenance Materials	--	--	5.64
Electric Power	kWh	$54.6X10^6$	1.46
Steam	Lb	$1,618X10^6$	4.87
Process Water	Gal.	$240X10^6$	0.24
Major Chemicals	--	--	4.42
Oper. Reagents & Supplies	--	--	1.24
Total Direct Operating Cost			25.17

Cost Per Ton of Ore	$14.67
Cost per Pound of U_3O_8	$ 7.48

4.3 Discussion

According to experimental data from the Department of Energy, Mines and Resources, hydrochloric acid leaching of low grade ore will extract thorium and radium to the extent that the residual radiation level in the tailings will be much lower than in existing milling operations and can meet the environmental criteria of 15 picocuries per gram. Uranium recovery as high as 98% has been demonstrated.

By regenerating hydrochloric acid from spent leach liquor, chloride ions are recycled to the leaching circuit and are prevented from discharging to the environment in large amounts.

Due to the low radioactivity of waste residues, the coarser fractions of these could be utilized for mine backfill and construction of tailings embankments. Approximately two-thirds of the waste residues would be suitable for these purposes.

Considering the plant environment, hydrochloric acid is corrosive and irritates the skin and lungs. However in most of the circuit the acid concentration is not high. In areas of high concentration the plant can be designed to be operated safely.

4.0 EVALUATION OF HYDROCHLORIC ACID LEACHING PROCESS

4.3 Discussion - Continued

Handling and transportation of concentrated radium concentrate could be hazardous but technology is available to design a safe system.

By recovering about 90% of the pyrite in the ore by flotation and roasting it to produce sulphuric acid, the sulphur content of the tailings can be reduced to the point where sulphuric acid generation in the tailings will be very limited and long term neutralization of tailings run-off water will probably not be necessary. Conventional sulphuric acid leaching plants could, of course, add to their circuits to accomplish the same purpose.

Comparing the economics of the hydrochloric acid leaching process selected for this study with conventional sulphuric acid leaching is beyond the scope of this study; however, there is no doubt that capital and operating costs for the hydrochloric acid leaching plant will be appreciably higher than the comparable sulphuric acid mill.

To make a meaningful comparison of the complete processes it would be necessary to consider the following in addition to the mill cost:

- value of mine backfill to the mine
- reduced cost of tailings ponds with hydrochloric acid leaching, due to the reduced tonnage of tailings
- the cost of radium chloride settling ponds which are not required if radium is recovered in the mill
- uranium extraction by hydrochloric acid leaching may be 2% to 3% higher than by sulphuric acid leaching

In addition, when comparing the leaching processes, facilities for pyrite flotation, roasting and sulphuric acid production should be either included or excluded from both processes. These facilities are not inherent in either process.

Lummus has recommended that bench scale test work on the hydrochloric leaching process be continued, particularly to provide more definitive data for cost estimates in the areas of hydrochloric acid regeneration and radium extraction.

5.0 EVALUATION OF CHLORINATION PROCESS

5.1 Capital Cost Estimates

Capital costs were estimated for a chlorination plant located in Saskatchewan. No specific site was selected and the costs do not include an allowance for construction indirect costs associated with a remote location.

Capital costs are estimated to be as follows:

	$ Millions
Installed cost of equiment:	
Grinding	3.6
Drying	1.0
Chlorination and chlorine regeneration	12.6
Hydrochloric acid and alkaline leaching	12.0
Uranium extraction as yellowcake	6.7
Extraction and separation of radium	2.7
Tailings disposal	0.1
Regeneration of hydrochloric acid	8.9
Sub-Total - Installed Equipment	47.6
Buildings:	17.2
Utilities:	2.2
Total Installed Plant Cost	67.0

5.2 Operating Cost Estimate

Operating costs do not include an allowance for costs incurred due to remote location such as camp costs and transportation of employees from settled communities.

5.0 EVALUATION OF CHLORINATION PROCESS

5.2 Operating Cost Estimate - Continued

Direct operating costs for the battery limits chlorination-leach plant are estimated to be as follows:

Item	Units	Annual Consumption	Cost $ Millions
Wages & Salaries	No.	(297)	7.10
Maintenance Materials	--	--	2.40
Electric Power	kWh	$23.7X10^{6}$	0.63
Steam	Lb	$581X10^{6}$	1.75
Process Water	Gal.	$95X10^{6}$	0.10
Major Chemicals	--	--	1.91
Reagents & Supplies	--	--	0.16
Total Direct Operating Cost			14.05

Cost Per Ton of Ore	$213.00
Cost per Pound of $U_{3}O_{8}$	$ 4.26

5.3 Discussion

Saskatchewan ores tested at CANMET contained radium 226 emitting radiation in the order of 5500 picocuries per gram. Laboratory results reviewed by Lummus indicated that the objective of 15 pCi/g in the tailings was not achieved by chlorination and leaching techniques. In fact, lowest radium levels in chlorination residues after leaching were in the order of 2000 pCi/g.

Uranium extraction by chlorination and residue leaching is quite high, in the order of 98% on one ore tested.

A major advantage of chlorination is that almost complete removal of arsenic is feasible. Arsenic trioxide probably could be recovered as a marketable product.

Nickel extractions by chlorination approached 100% on two ores tested.

By regenerating hydrochloric acid from spent leach liquor, chloride ions are recycled to the leaching circuit and are prevented from discharging to the environment in large quantity.

Although processes employing hydrochloric acid and chlorine are potentially hazardous, safe operation can be attained by proper plant design and by employing well trained operators. Fluid bed chlorination is well proven in other metallurgical operations, for example, titanium.

5.0 EVALUATION OF CHLORINATION PROCESS

5.3 Discussion - Continued

The chlorination process is obviously an expensive one but it may be justified if the radioactive objectives can be met. Lummus have recommended that priority be given in further testing toward substantial improvement in radium extraction.

ACKNOWLEDGEMENTS

The author would like to thank the management of Lummus Canada Inc. for permission to publish this paper. We also wish to thank CANMET engineers and scientists, particularly W.A. Gow, G.M. Ritcey, Dr. K.E. Haque and Dr. J.M. Skeaff for their invaluable assistance with the study. The major role played by Dr. G. Angelov in the study is also acknowledged. Dr. Angelov was employed with Lummus when the study was carried out and is now retained by Lummus.

DISCUSSION OF PAPER BY I. EDWARDS (CHLORINATION)

J. Goode	On the high grade ore, Th^{230} would grow in. What do you intend to do about that?
I. Edwards	Although it was not cited on the flowsheet, it would be removed as in the low grade ores, probably by SX or IX.
E. Krause	What is the chlorination temperature?
I. Edwards	It is in two stages, at 250-300°C and 500-550°C.
E. Krause	Did you meet the liquid specifications on Ra^{226}?
I. Edwards	No, there is no viable process for the absorption of radium from chloride solution.
V. Lakshmanan	Is the operating cost based on northern Saskatchewan (for Cl_2 route)?
I. Edwards	Yes, that was taken into account.
J. Fisher	Where would you put the radionuclides after isolation?
I. Edwares	Disposal for burial.
L. Heyman	Were there any chloride analyses on the tailings?
I. Edwards	No, there was no balance made

Energy, Mines and Resources Canada
Énergie, Mines et Ressources Canada

CANMET
Canada Centre for Mineral and Energy Technology
Centre canadien de la technologie des minéraux et de l'énergie

CANADIAN PARTICIPATION IN THE NEA/IAEA WORKING GROUP ON URANIUM EXTRACTION

M.C. Campbell
Extractive Metallurgy Laboratory

May 1981

For presentation to the Canadian Uranium Producers Metallurgical Committee in Saskatoon, Saskatchewan, May 19, 1981

Project ERP-4.4.0.2.01
Uranium Extraction - Conventional Technology

ENERGY RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
DIVISION REPORT ERP/MSL 81-51 (OP&J) DRAFT

CANADIAN PARTICIPATION IN THE NEA/IAEA WORKING GROUP ON URANIUM EXTRACTION

by

M.C. Campbell*

ABSTRACT

The author attended the fourth meeting of the NEA/IAEA Working Group on Uranium Extract (WGUE) in Paris in February 1981. This report presents a description of the role in the WGUE and a summary of the Paris meeting to inform the Canadian uranium industry at the annual meeting of the Canadian Uranium Producers Metallurgical Committee (CUPMC). The WGUE is seen as an international equivalent of the CUPMC.

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*Manager, Extractive Metallurgy Laboratory, CANMET, Energy, Mines and Resources Canada, Ottawa.

INTRODUCTION

In October 1978, the NEA/IAEA Working Group on Uranium Extraction (WGUE) was officially formed at the request of the FCC (Committee for Technical and Economic Studies on Nuclear Energy Development and the Fuel Cycle) of the Nuclear Energy Agency (NEA). The NEA is an agency of the OECD (Organization for Economic Cooperation and Development) and is located in Paris.

The OECD represents only 14 developed industrialized countries including Canada. In order to involve other uranium producing countries, principally South Africa, the participation of the UN agencies, the International Atomic Energy Agency (IAEA) located in Vienna was requested. Thus a joint NEA/IAEA working group was established.

The terms of reference for WGUE were presented at the 1979 CUPMC meeting. Basically, the WGUE is a medium for information exchange on an international basis - almost an international CUPMC.

Canada was asked to send a representative and I was appointed by the Department of Energy, Mines and Resources to be that representative. There have been four meetings of the WGUE beginning in Vienna and then alternating between Paris and Vienna. The availability of facilities for meetings, translations and interpretation, and printing dictates that meetings be at the NEA or IAEA headquarters. Furthermore, the NEA and IAEA provide permanent secretaries for the WGUE. The former NEA Secretary was Bruce Boyd, a Canadian on leave of absence from EMR. Currently, Dr. Derek Taylor, a British geologist is the NEA man. Frank Hartley, an Australian from AMDEL was the IAEA representative. Currently Don Seidel from the USBM, Salt Lake City is the IAEA Secretary. These people provide both coordination and expertise.

The Chairman of the WGUE has been Henry James formerly with the Atomic Energy Board of South Africa and now Vice-President of the National Institute for Metallurgy in South Africa. I was the Vice-Chairman of the WGUE. At the 4th meeting, I was elected Chairman and Georges Lyaudet of Cogema in France is the Vice-Chairman.

I see a close parallel between the objectives of the CUPMC and those of the WGUE. I see the CUPMC and its members as the principal Canadian contributors and beneficiaries of the Canadian participation in the WGUE. Therefore, I thought it important to report to you on the objectives and activities of the WGUE, and seek comments and feedback from you.

The Chairman (Mr. James) introduced participants (see list as Appendix I). He emphasized that attendance is voluntary so that the membership is really the same as the attendance. He made a point of having each country encourage fuller participation by the corporate sector. I therefore extend to each of you an invitation to attend WGUE meetings. The next meeting will be in Vienna in February. If you can convince your management (as I must convince mine) to send you, you are most welcome.

STATUS REPORTS ON NATIONAL RESEARCH PROGRAMS

Each participant gave a brief resume of R&D activities in their country. These were presented in the order of seating so that there is no particular pattern.

COMMISSION OF EUROPEAN COMMUNITIES

The CEC has analysed the state of R&D activities in the member countries. The principal interest is in the recovery of uranium from the limited low grade resources available in Europe to maintain a secure supply at an acceptable price. Research topics encompass studies of technico-economic feasibility, and process technology. Feasibility studies are underway on recovery from phosphoric acid liquors, extraction from sea water, and the prevention of pollution.

Extraction techniques are being developed under projects on:

- in situ leaching of small deposits
- bacterial leaching
- high temperature, high pressure leaching of sandstones and shales
- recovery from refractory ores from Greenland and calcines from oil shales and lignites
- extraction from phosphatic sediments
- extraction from phosphate rock treatment wastes
- extraction from phosphoric acid liquors
- extraction from sea water.

The CEC has no laboratories so that some 2.4 European Units of Account are being spent annually on contracts. Performers are organizations such as the Warren Spring Laboratory. There have been no publically available reports arising from the work to date.

SWEDEN

Sweden generates 30% of its electricity using 6 nuclear reactors with 6 more to start up in the future but has no uranium producers. LKAB had been contracted to locate a uranium mine and has done so in northern Sweden but must now convince the local community of its value to them. An environmental assessment document was prepared and is under consideration. This is a public document but is available only in Swedish.

The Ranstad plant, which has been in stand-by condition since 1969, has been maintained as an R&D facility. However R&D funding expires at the end of 1981 and a severe cutback in activities is expected. The Ranstad project was designed to recover all values from the Ranstad shales. A 50-100 liter/hour pilot plant was completed and a 7 tpd leaching plant has been operating. A gasification pilot plant was also completed but is now being used to test wood and peat.

UNITED KINGDOM

In the U.K., the Warren Spring Laboratory is the principal performer with support from the CEC and the Mining Requirements Board. An assessment of the potential for uranium extraction from phosphate liquors is being prepared and an the inventory of technology important in the European situation has been completed. A paper was presented at the Cambridge, Mass. meeting on recovery from sea water on the operation of a small pilot plant in UK waters.

The Uranium Institute Meeting in London, September 2-4, 1981 was cited as an opportunity for the presentation of technical papers.

ITALY

Mr. Cordero, the major participant from Italy, was not in attendance during this part of the meeting so very little information was available. The local authorities have yet to give permission for the startup of Italy's first uranium extraction plant at Novazzo. Italy has done some work on recovery from sea water, but sees engineering, not chemistry, as the principal problem. At earlier meetings, it had been reported that research was poorly coordinated in Italy. Most of the research was performed by Agip, the company controlling the only potentially exploitable uranium deposit. Agip is basically a petroleum company so that uranium is a minor aspect. They have sponsored a study on the Novazzo ore by RTZ Ore Sorters and on bacterial leaching. Italian phosphates are high in uranium and work on recovery from phosphoric acid has been supported. Agip also has involvement in uranium in Nambia, Niger, Australia (Mt. Isa) and Canada (Bancroft) through ownership in several ventures.

FRANCE

France produces 30-35% of its electrical needs from nuclear plants. Therefore, uranium is an important element in industrial policy.

The newest development is the new mine at Lodeve in the south of France. This operation has a capacity of 900 tonnes of uranium a year. Recovery of molybdenum is a possibility in the future.

France is heavily dependent on operations in Niger and Somalia. They import some 1900 tonnes of uranium a year from these countries and play a major role in the development of technology.

Currently, R&D activities in France by Cogema are focused on uranium recovery from phosphoric acid. A feasibility study some 2 1/2 years ago supported a plant. Pilot plant work has proceeded, however the basic decision is a political one. France is also working on the development of specific IX resins to alleviate environmental problems. Mr. Lyaudet will provide the Secretariat with a list of patents, references and conferences. This will be included in the summary record of the meeting.

The R&D budgets in France are being maintained principally aimed at improving process economics. A report on preconcentration was given at Vichy in 1980. The reference will be provided via the Secretariat. Copies of 5 reviews in French were provided on; IX Resins, Acid leaching, the Lodeve mine vein type deposits in France, and the Cogema pilot plant. The English translation will be available soon from the NEA and sent to the various participants.

USA

Mr. Jones gave perhaps the most pessimistic report. Recent consultations with industry representatives provided a gloomy outlook with the price for uranium dropping from $44 to $27 a pound and producers having problems covering costs. Mine employment has dropped perhaps 20% (3,500 to 4,000 out of 16,000 to 20,000 employees). There have been no shutdowns of the 40 plants in the U.S. but several are in standby status. In spite of this, 1980 had a record production of 21,000 tons of U_3O_8. However, a substantial drop (perhaps 15-20%) is expected in 1981. Anaconda (Grants) has stopped mining and is working on stockpiled ore.

In this environment of declining price, several new plants are scheduled to come on stream: Energy Fuel Nuolear in Utah, Union Oil in Wyoming (they had a disastrous fire 3 days after startup); Homestake, Pioneer Uravan, Conoco at Grants, and Gulf in the Taylor district (New Mexico?). Three more phosphoric acid plants are also slated for startup.

In the U.S. virtually all R&D is performed by the private sector with their own funds or under government contracts. Performers include:

i) Bendix Field Engineering is working on the Chattanoga shales. This work is well documented in public reports available from the U.S. Department of Energy.

ii) Oak Ridge National Laboratories are working on reducing solvent losses from phosphoric acid plants using radioactive tracers, ^{32}P for TOPO; ^{14}C for the diluent.

iii) MIT is working with Rohm and Haas on resins for recovery of uranium from sea water at Woods Hole, Mass. in a $120,000 project.

iv) USBM (Salt Lake City) is performing amenability tests on intermediate grade (100-500 ppm U) carbonaceous shales from Red Desert, Wyoming on a $75,000 budget.

Mr. Jones commented that an increasing proportion of U.S. uranium production is coming from solution mining in Texas. Ten operations produce 3,000 tpy (15% of US production). Ammonium carbonate is being supplanted by CO_2 and O_2 because of concern with ammonia ion excursions influencing ground water quality. With proper pH control lime deposition is avoided. O_2 being used exclusively, trials with peroxide did not indicate economic advantages.

JAPAN

Dr. Hirono read a review of "Topics in Japan" from a prepared text. I have a copy on file. The topics included:

i) Uranium leaching from autunite ore. Sulphuric acid leaching with ferric sulphate followed by IX gave 95-97% uranium recovery.

ii) In situ leaching of a zeolite like mineral (heulandite) in an alkaline solution injected into bore holes.

iii) Uranium recovery from dam water.

iv) Uranium recovery from sea water.

Two papers by M. Kanno which had been presented at the Cambridge, Mass. conference in December 1980 were provided. The basic process involves adsorption on titanium oxide, elution and recovery by steam stripping or electrodialysis followed by IX or ion flotation. Costs are estimated at \$1436 and a hypothetical best cost of \$475/lb U_3O_8.

CHILE

Chile has no uranium ore body but has a plant (Compania Minera Sagasca) which recovers about 50,000 lbs U_3O_8/yr as a byproduct from a copper leach solution.

Research has progressed in the development of a new IX (Vanel) column design for Chuquicamata (?) in which there is no need for resin transfer. This design is being considered for a resin-in-pulp installation. The IX column was developed because resins tended to float in high specific gravity solutions . A patent application may be made. A copy of the schematic diagram is attached as Figure 1.

CANADA

A report "An Overview of Uranium Extraction Research at CANMET" ERP/MSL 81-20 (OP) was presented. This report was well received and the Chairman held it up as a model for others.

In addition comments were made on the general status of the Canadian uranium industry and on related acitivites in Canada such as conferences and seminars.

AUSTRALIA

Dr. Smith talked from information received from Dr. Alfredson who was not present. A review of projects was given.

i) Alligator River - a report has been completed by the Office of the Supervising Scientist for this project on heavy metals, flora and fauna as a baseline study.

ION EXCHANGE COLUMN

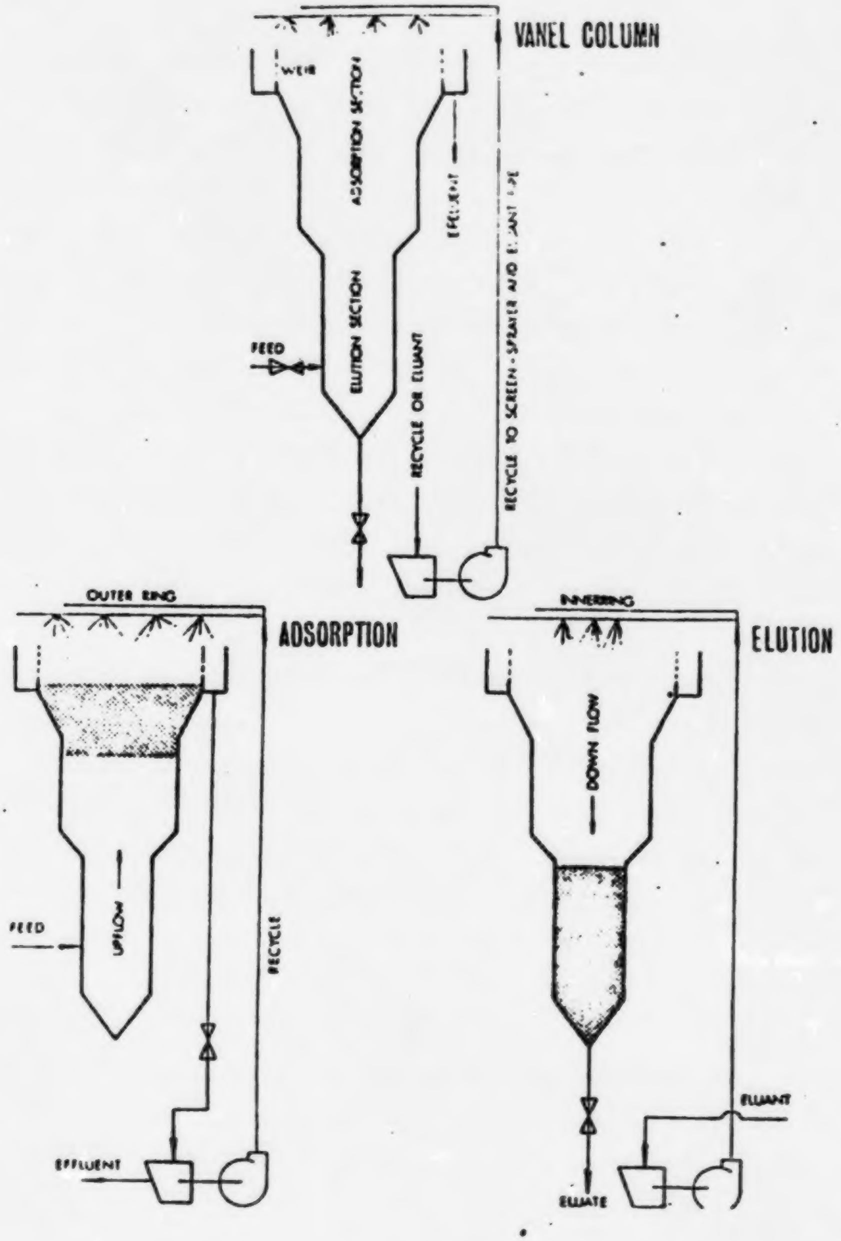

FIGURE 1

ii) Yeelirrie has been delayed and may not start up. The Western Mining is looking at Olympic Dam (Roxbury Downs) as an alternative project for the company. The Olympic Dam copper property is only in the exploration stage.

iii) Jabiluka and Koongarra (in a national park area) are likely to receive approval but there is a delay due to foreign ownership (Noranda Australia Ltd.). It may be that market conditions will affect these.

iv) Honeymoon is a solution mining prospect on which R&D is now being done.

Australia has no nuclear power plants since they are uneconomic in the face of abundant coal. Plants could possibly be sited in Western Australia and the Northern Territory but the demand is small and the grid is not extensive. Uranium and nuclear power are hot political issues and the Socialist party is strongly anti-nuclear. An enrichment plant has been considered and a feasibility report will be available in several months.

Dr. Smith was confident that an economic recovery (growth rate of 3-4%) in Australia was imminent. Many major projects are being considered totalling $3 to 4 billion. Aluminum plants are the main thrust because of cheap energy so that an enrichment plant may have low priority. Liquefaction of coal is another area for development. The demand for electrical energy is expected to double in ten years and it is possible that nuclear will have to play a role.

ARGENTINA

In Argentina R&D is aimed at improving uranium recoveries. They are shutting down two uranium plants, one because the economics of recovery from 600 ppm ore are inadequate, the other because reserves have been depleted.

Two new mills are under construction:

i) Los Ciantas in Cordero province will start in 1981 to produce 100 tonnes U_3O_8 a year from an open pit, heap leaching operation on 300 gram/ton ore and acid consumption of 15 kg/t is predicted.

ii) Sierra Pintata is a 500 t/yr operation to start up in 1982. Exploration is continuing with $2 million drilling in 1981. However, everything is moving and construction has started.

The uranium plant at San Raphael is using heap leaching on material with grades of 200 to 800 g/t. High grade material is sent to Malague for conventional leaching. Recoveries at San Raphael are about 75% in 4 meter high heaps.

SOUTH AFRICA

Two new plants are being actively planned. West Rand has announced a uranium recovery plant with a gold mine but no capacity has been given. The Kiroo project has slowed down because of the market condition.

R&D activities at the Atomic Energy Board are sponsored partly by industry. Some work is being conducted on resin-in-pulp application for low grade reclaimed tailings using heavy resin in an upflow-column at 10% solids. Solvent-in-pulp is also an area of interest.

AEB has undertaken an integrated program on all aspects of Vadesmum (?) ores. This includes a literature review using a computerized information processing system, qualitative and quantitative mineralogy, leaching fundamentals of uraninite and brannerite (electrochemical mechanisms, acid concentration, oxidants, etc.), kinetics of leaching, and hydraulics of mixing (air vs mechanical). Work is also underway on the behaviour of thucholite. This includes the effects on grinding, liquid-solids separation, and leaching. There is a shift in emphasis in programs away from process toward more basic understanding of the chemistry.

The optimization of belt filter performance is a subject of study. Performance has not been up to expectations with soluble losses several times higher than with conventional filters. Operators have found that by reducing throughput these losses can be lowered. Disc filters are being studied as an alternative.

NIM and AEB are jointly sponsoring a vacation school on uranium extraction in Johannesburg the last week in July 1981. Most of the notes from the vacation school will be made available for the NEA/IAEA WGUE book on uranium extraction.

AEB has not been affected by the uranium market. There are about 20 technical personnel and a total staff of 30 people. NIM does some work on uranium so that the total uranium effort would be about 25 py.

Vaal Reefs are building a pressure leaching plant of the Sherritt-Gordon type based on very encouraging results on a pilot plant scale. Vaal Reefs is expected to add a full scale pressure leach circuit to the existing three leach plants within two years. They are going to scale up directly from the pilot plant. Other ores, (e.g., Afrikanner Lease) have also been tested at the Vaal Reefs pilot plant. The Lurgi pipe reactor was tested on a South African ore. No data were made available but it was described as an interesting possibility.

GERMANY

In Germany there are three principal streams of R&D on uranium; government subsidized work in companies, corporate R&D, and R&D by machinery manufacturers (Lurgi and Humboldt).

i) Uranium from sea water is being studied by Uranertz in conjunction with the German Nuclear Power Co. Laboratory work has now been extended to a pilot model in the Gulf Stream using a float system. A self propelling adsorption column is also under development in which 30-50% of the uranium in the sea water is absorbed. A paper was presented at Cambridge, Mass. in December 1980. Costs are being reduced and $200 a pound seems achieveable (in conflict with the Japanese figure - Dr. Floter missed the Japanese presentation).

ii) Phosphates with high Al_2O_3 seem to have more uranium. Some research is aimed bacterial leaching and the use of yeasts to recovery uranium.

iii) A new treatment process for calcrete ores uses cycloning to decrease water consumption. The coarse size is dewatered on belt filters, slimes with thickeners and the overflow is recycled. Patents have been applied for and a publication will follow later this year.

iv) A process of biological assisted leaching with controlled oxidation is to be published by Uranertz in 1982.

v) In situ leaching with CO_2 and O_2 is being tested in Wyoming.

vi) Radium fixation with $BaCl_2$ using a "floc building" step between ponds is under investigation.

vii) In situ analysis of uranium deposits using an energy dispersive analyser with a Cobalt 60 source. This portable field unit is fed with a 2 gram sample. Uranertz is not satisfied with the current detection limits below 100 ppm U.

viii) Acid pressure leaching in the Lurgi pipe reactor provides space saving and high throughput. It is good for high temperature alkali leaching but insufficient data are available on acid circuits. There seems to be a question mark with respect to oxidant supply to the system. France rejected the system for Lodeve ore. South Africa is piloting a minimum sized plant but data will be available only in several years time.

ix) SX work has been focussed on Krebs unit to minimize crud formation.

SPAIN

The Spanish delegate Dr. Josa did not attend because of the attempted coup in Madrid two days before the WGUE meeting. However, I visited Dr. Josa in Spain the following week. Briefly, Spain has a difficult energy situation with no oil, no gas, no good quality coal and little uranium. They have 3 nuclear power plants with a total capacity of 1000 MWe. Seven new plants are under construction and three more have been approved. When completed they will generate 10,000 MWe.

Spain exploited their limited uranium reserves as insurance against rising international prices. Current production amounts to about 360 tonnes of U_3O_8 a year from three ore-based plants and one phosphoric acid treatment plant. A further 80 tonnes is available from phosphoric acid.

I visited the ENUSA plant near Ciudad Rodrigo in Salamanca province. This 135 tonne a year plant uses heap leaching and solvent extraction. It is a well designed and operated plant, but my impression was that it has been heavily subsidized.

Research in Spain is conducted mainly at the Junta Energia Nuclear in Madrid (Dr. Josa is the Chief). Projects are aimed at improving existing processes (leaching using $FeCl_3$ and solvent extraction), examining options for expansion (preconcentration and IX), and extraction from unconventional sources (fluid bed combustion of lignite and extraction of the 200 ppm uranium).

PUBLICATION ON URANIUM EXTRACTION TECHNOLOGY

In 1978 and 1979, the IAEA in cooperation with member countries and with a final discussion at the WGUE prepared a publication "Significance of Mineralogy in the Development of Flowsheets for Processing Uranium Ores". This was published by the IAEA in 1980 as Technical Reports Series No. 196 and is available from UNIPUB, 345 Park Ave. South, New York 10010. It is a fairly basic book but a valuable addition to a library.

One of the tasks undertaken by WGUE is to update the data in the IAEA publication and to provide more insight into uranium extraction practice, unit operations and research. A new publication is scheduled for late 1982 by the NEA with the WGUE acting as editors. The publication will include a section on uranium extraction in each of the participating countries, and sections on the principal unit operations and related research. I will be preparing the Canadian input. Many of you have already given me material for this.

In the near future I would like to send a copy of our submission for your comments and corrections, as necessary.

OTHER MATTERS

SEA WATER

A proposal was made to have a separate sub-group on sea water. It was agreed that such a sub-group already existed although not as part of the WGUE. The interested countries, U.S., Japan, Germany and the U.K. already have their contacts. The WGUE can be kept informed by participants from these countries as a regular agenda item.

The challenges were outlined at the MIT meeting in December 1980. A paper by A.D. Klymecs of Oak Ridge gave a good account of the status of uranium recovery from sea water at the MIT meeting. The proceedings should be available from MIT in the near future. The challenges are:

i) Absorbers - high grade TiO_2 (HTO) gives a 300 to 1 concentration on elution but is somewhat soluble in water and leads to high costs.
ii) Ocean studies - warm water is necessary for high capacity absorption.
iii) Improved cost analyses and lower cost systems. Costs are largely unknown. Costs of $300-900/kg U are hardly an economic reality. A one tonne a year pilot plant would be necessary to develop an acceptable basis for costs. All work to date has been on synthetic systems. Germany and Japan have targeted for a 1000 tpy plant by 2000.

Basically there are no real breakthroughs, nothing that will upset supply and demand. Exxon did a study that had very negative conclusions. The MIT meeting identified problems and focussed on R&D goals. A $200 a pound ultimate cost is foreseen. There is no great international enthusiasm, only curiosity.

COORDINATION WITH MILL TAILINGS GROUPS

Within the NEA/IAEA setup there is a nuclear waste group to which a tailings advisory group reports. It was agreed that the WGUE would cooperate with the tailings groups by offering to organize a workshop on uranium extraction options which would help alleviate environmental problems. Options suggested were:

- sand/slimes separation
- separation of radium and chemical sludges from tailings
- sulphide separation (flotation)
- use of novel flow sheets (nitric acid digestion).

It was agreed however that the WGUE must not get embroiled in tangential activities such as criteria for long term storage.

ACTIVITIES OF OTHER WORKING GROUPS

This was mainly a discussion of the respective roles of NEA and IAEA committees and working groups and their relationship to the WGUE. The WGUE should examine extraction technology and qualitative economics. Uranium extraction technology cannot be accepted as always being static, known and available. WGUE provides a forum for discussion of alternatives, sharing of experiences and provision of advice to the FCC. Geological occurrence does not mean uranium reserves if the uranium is not extractable at a reasonable cost.

Manpower availability is an important aspect in uranium availability. WGUE can help in the task of technology dissemination and aid in the task of training and development of personnel.

NEXT MEETING

A meeting of the executive group has been proposed for Paris in September 1981 to discuss the NEA publication. The next WGUE meeting was proposed for early 1982 in Vienna.

The WGUE also recommended a large meeting or symposium on uranium ore processing in late 1982/1983. Attendance of about 100 would be expected.

COMMENTS

The WGUE is an interesting group. The value to the participants has been questioned, however those attending the Paris meeting were very appreciative of the opportunity to discuss areas of mutual interest with representatives from other countries. The WGUE seems to be now developing its own character and a good working relationship between participants is arising. It was as a result of the Paris meeting that I saw the parallel with CUPMC.

I would like to encourage a dialogue with CUPMC members on the WGUE. As Chairman I will be in a position to influence the committee and I would like to influence it to Canada's benefit. I welcome any comments and I would certainly welcome attendance at future WGUE meetings by company representatives. The next meeting is in Vienna in early 1982, perhaps an avid skier could swing the trip.

APPENDIX I

LIST OF PARTICIPANTS

Country (alphabetical in French)	
South Africa	Mr. Henry E. James, Vice President National Institute of Metallurgy
	Dr. David W. Boydel, Extraction Metallurgy Div., Atomic Energy Board
Germany	Dr. Wilfred Floter, . R&D Division, Uranertzbergeau GmbH
Argentina	Mr. Roberto Costarelli, Raw Materials Div., CNEA
Australia	Dr. Robert Smith, Counsellor-Atomic Energy, Australian High Commission, London
	Mrs. Vanessa Fanning, First Secretary Permanant Delegation - NEA
Canada	Mr. Michael C. Campbell, CANMET
Chile	Mr. Elias J. Andalft, Technical Advisor, Atomic Energy Commission
France	Mr. Georges Lyaudet, Cogema
Italy	Mr. Fabio Pantenetti, CNEN
	Mr. U. Cordero, Agip/SPA
Japan	Dr. S. Hirono, PNC
United Kingdom	Mr. G.I.W. Llewelyn, Economic Programs Br., UKAEA
	Mr. A.T. Triggs, British Civil Uranium Procurement Organization
Sweden	Mr. G. Olsson, Ranstad Skifferaktiebulag

U.S.A.	Mr. Jack Q. Jones, U.S. Dept. of Energy (Grand Junction)
CEC	Mr. M. Donato, R&D Programme on Raw Materials and Uranium
IAEA	Mr. Don Seidel, Joint Scientific Secretary
NEA	Dr. Derek M. Taylor, Joint Scientific Secretary

DISCUSSION OF PAPER BY M. CAMPBELL

A. Moorish	Are the belt-filters in South Africa used in production?
M. Campbell	Yes, no reports are available.
G. Ritcey	Are there any dicussions on tailings in the Processing Committee?
M. Campbell	No.

NATIONAL RESEARCH PROGRAM

ON

URANIUM TAILINGS MANAGEMENT

V.A. Haw

February 15, 1982

NATIONAL RESEARCH PROGRAM
ON
URANIUM TAILINGS MANAGEMENT

Introduction

What I have to say today is simply an account of events connected with the development of what has become known as the "National Research Program on Uranium Tailings Management." This is, of events up to this point in time. A report on this program has now been prepared by a committee known as the "National Technical Planning Group," which contains descriptive material on uranium mining operations, and particularly the existing practice of uranium tailings management. It also proposes a structured research program covering a period of three years, with the ultimate objective of developing methods for the safe disposal of uranium tailings so that their further management is unnecessary. It is this report of the National Technical Planning Group that I want to tell you something of this morning and also to provide you with some background on why the report was prepared.

Background

Before going any further I would like to make a point here that has come up time and again in discussions with industry in particular, and that is,"Is the long-term management of uranium tailings a problem?" In one sense, this is what the research program is all about -- to determine the nature and extent of the problem, or in the minds of some, is there a problem at all. In any event, whether a problem does exist, there is certainly a perceived problem in the minds of the public, and this being the case there is concern at the political level which must be addressed.

In the fall of 1979, CANMET took the initiative to organize a meeting of potential performers of uranium tailings research, the first step towards the establishment of a comprehensive and integrated national program. The meeting took place in Ottawa on January 14, 1980. It brought

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together those that have been involved in sponsoring and performing research on the problem -- the mines that produce the tailings, the federal government departments that set regulations and standards, and the research groups including provincial research organizations, universities and consultants, all of which will be needed to perform research and studies to help define the problem and propose solutions for the management of uranium tailings.

As a result of this meeting, a decision was made to form a National Technical Planning Group representing different organizations, public and private, which were concerned with uranium tailings management, to examine the situation and make recommendations. Twelve members were selected for the group, representing the federal government, the provincial governments of Ontario, Saskatchewan and Newfoundland, industry, provincial research councils, and universities. The approved terms of reference of the National Technical Planning Group were:

> The Group will be concerned solely with research related to the long-term abandonment of uranium mine tailings and will:
>
> (1) review present activities and sources of funding,
>
> (2) propose a research program structure defining priorities and objectives,
>
> (3) estimate a program schedule, cost and cash flow, and
>
> (4) propose a program management structure.

After selecting the Group members, further discussions between the federal government and the provinces made it clear that a federal/provincial steering committee would be necessary to ensure that the interests of both levels of government were fully considered. The steering committee met periodically during the course of the work of the National Planning Group, monitoring progress and finally approving the conclusions and recommendations of the Group's final report.

The first meeting of the Planning Group was held in April, 1980, and its final meeting was held early in March, 1981, to give its collective approval

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of the report that had been prepared and coordinated by the Chairman of the National Planning Group, Dr. Philip A. Lapp. In the interval, which lasted about a year, the Planning Group as a whole held meetings with provincial representatives of departments in Ontario and Saskatchewan at which the activities of the various departments and agencies at the provincial level were described in some detail insofar as they related to uranium tailings management. The Group also visited a number of uranium tailings areas, both active and inactive, where they had the opportunity of discussing with company representatives their approaches to tailings management. While at Elliot Lake the Group met with scientists from CANMET laboratories who are active in this activity and have a number of experimental sites in the area in which they are cooperating with local companies in a number of projects. Visits to provincial research councils of Ontario and Saskatchewan were also included in the Group's itinerary where they had the opportunity of seeing facilities available for conducting research on uranium operations and were able to hold discussions with staff.

The National Technical Planning Group completed its field visits and meetings with outside groups last October, and spent from October to March analyzing its findings and preparing contributions to specialized areas for inclusion in the final report. An inventory of research on all aspects of uranium tailings storage and disposal was made and presented in the form of a matrix: one dimension showed the type of work performed - containment source, mining, milling, tailings management and pathways analyses; the other listed the performers. When the material had been digested, the Group concluded that: there is a lack of data to determine the essential characteristics of ground and surface water pathway regimes from specific sites into the biosphere; air and food chain pathways are not yet fully understood and insufficient effort is devoted to predicting contaminant movement along pathways over the very long time spans associated with the half lives of the radionuclides. In other words, a broadly based national research program on uranium tailings management is needed. A decision was made to recommend a research program consisting of three main activities: modelling, measurement and disposal technologies.

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A report was prepared by the National Technical Planning Group wherein various members contributed sections dealing with their own sphere of specialty. These were produced as appendices which were coordinated by the Chairman of the National Technical Planning Group, Philip A. Lapp, the consultant retained by CANMET, who also wrote the main part of the report based on the various members' contributions. I might add that it has gone through several revisions and is now ready for publication, but its release will be delayed until the three governments concerned have officially received it and are prepared to take action on its recommendations.

Report Content

I will attempt to touch today only on a few highlights of the report, as it is quite extensive - more than 135 pages. Much of it deals with subject matter that would be rather commonplace to an audience such as this.

The report takes full cognizance of the importance of the uranium mining industry to Canada, and particularly to the provinces in which mining operations are located. It also recognizes that whereas Canadians derive major benefits from uranium mining, there are environmental and health consequences of the associated mining and milling operations that are not yet fully understood. It points out that there are already over one hundred million tons of mill tailings on the surface that need attention. About one-third are now inactive, while the remaining tailings are associated with operating mills where pollution abatement procedures are being followed. At current production rates, approximately 14,000 tons are being added daily, which I understand will about treble the amount of tailings presently on the surface by the end of the century. The report describes in some length the mining and milling operations at uranium mines and also the physical and chemical properties of uranium tailings. I think we can pass over this part, which I think will be of little interest to you. Remember that this report is directed primarily to federal and provincial government officials who have the responsibility of assessing the contents and the recommendations for support of a research program, and many of these people are laymen insofar as the uranium mining industry is concerned. The report will also be available to the public in general, therefore, an element of education is required to inform people of the basic facts on uranium mining operations.

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It is perhaps worthwhile to quote almost literally from the report a statement of the fundamental issue which is treated under the sub-heading "After Shutdown - A Statement of the Problem:"

> "When a mine is depleted and the mill eventually shuts down, the tailings are immediately affected. With no effluent from the mill, the watertable in the tailings basin begins to drop, and tailings near the surface become dry and more readily blown by wind away from the management area, unless covered and stabilized in some fashion. There will be some contingent seepage of groundwater out of the tailings and surface water runoff, resulting from the precipitation and other natural surface flows.
>
> "Some major environmental concerns associated with the shutdown are:
>
> (1) contamination of surface water or groundwater systems with contaminated seepage or leachate;
>
> (2) redissolution of radium from barium-radium sulphate sludges in treatment ponds;
>
> (3) tailings impoundment dam failures;
>
> (4) emissions of radon gas and direct radiation from dried-out tailings area surfaces;
>
> (5) the contamination of soil and vegetation from wind-blown radioactive dust particles from dried out tailings impoundments; and
>
> (6) physical removal of the tailings by man."

While active management schemes are used to maintain the emissions from inactive tailings within regulatory limits, the steps that may be required for the safe total abandonment of uranium tailings are not yet known. The problem then is to devise such steps based on a better understanding of the mechanisms of contaminant transport and dispersion, and to throw light on how inactive tailings can be treated for the safe abandonment requirements of either/or both levels of government.

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The report goes on to describe some of the current research and development related to tailings abandonment. It points out that there are five broad strategies being examined for the final disposal of uranium tailings:

(1) surface, (2) shallow burial, (3) deep burial, (4) deep lake disposal, and (5) removal of the contaminants in the mill process.

It is recognized that these strategies are not fully applicable to all uranium producing regions. The aim of each is to identify feasible methods of closing out tailings areas, which result in acceptably low adverse effects on the environment and man. Research activities that are common to all the disposal strategies listed in the report are:

(1) measurement and study of the physical and chemical properties of uranium mill tailings;

(2) monitoring the content, migration and distribution of radionuclides and heavy metals in abandoned and active tailings over time, as well as in the associated ground and surface water regimes;

(3) study of the leachability of radioactive components in waste rock and tailings;

(4) analysis of pathways whereby contaminants may reach the environment and man;

(5) hydrogeological studies of tailings management areas;

(6) chemical and physical stability of barium-radium sludges;

(7) study of the uptake of radionuclides by vegetation on tailings and in the vicinity of tailings areas;

(8) research on the effect of bacteria on release or retardation of radioactive effluents;

(9) improvements in sampling and analytical methods;

(10) sorption of radionuclides by natural materials.

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Notwithstanding the considerable amount of research effort that has been completed to date (the total accumulated research on uranium tailings management to date amounts to about $7.8 million), there has been a lack of central coordination of research on tailings. This means that each sponsoring organization has focused, understandably, on its own areas of responsibility and interest. Fortunately, because of the standing committees and panel that exist for exchange of information, duplication of effort has not been excessive. However, data and results have been generated that cannot be compared due to lack of agreement on analytical standards and procedures across the country. The planning group was also struck by the lack of sufficient data to determine the essential characteristics of ground and surface water pathway regimes at specific sites, with the exception of the Nordic tailings where research is presently being conducted. Air and food chain pathways are not yet fully understood. Moreover, there has been insufficient effort devoted to the prediction of contaminant movement along pathways over the very long time spans associated with the half lives of radionuclides. Even over the short term, there have been an inadequate number of measurements made on contaminant concentrations outside the waste management areas of either active or inactive tailings. Indeed, on the basis of prima-facie evidence revealed by measurement alone, there are not enough data to determine the extent of the long-term problem.

In light of this information and the conclusions reached by the National Technical Planning Group, a national research program has been proposed, which I will now briefly describe.

1. Modelling Program. The objective of this program is to provide a forecasting tool for predicting the long-term behaviour and dispersion of contaminants within and beyond the waste management area. Models will be used to identify and compare the long-term effects and efficacy of current and proposed disposal alternatives. They also are required as a tool in deriving appropriate standards for managing uranium waste. The modelling program will specify the critical field measurements needed to develop the models and later to verify them.

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The principal program activities under modelling fall into four categories:

(i) model and development and validation;

(ii) data base management;

(iii) computer program development; and

(iv) interpretation, assessments and program direction.

2. National Measurement Program. The objectives of this program are to provide the basic data needed for model building, to verify models that have been constructed and to establish a national uranium waste measurement data base. The data base could become the standard rererence for regulatory agencies and uranium producers, and should be developed to uniform national standards through federal/provincial/industrial agreement. It is intended under the national measurement program to choose three representative tailings sites on which detailed measurements will be made. This part of the work will comprise four parts:

(i) sample design, installation and implementation;

(ii) analyses of samples:

(iii) analytical, mineralogical and other research under controlled laboratory conditions; and

(iv) data base management.

The sampling program will be designed to yield hydrological and hydrogeochemical data needed by the modelling program. The Group also believed that initial research work into improving analytical techniques and instrumentation was required and that there was considerable scope for new concepts in sampling methods.

3. Disposal Technologies Program. The objective of this program is to identify economically feasible methods of closing out tailings areas which will result in acceptably low adverse effects on the future environment and man. The work plan under this heading is not as well defined because, as the report states, the definition of the problem must be more fully established by input of hard data from the national measurement program.

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For this reason, resources devoted to research on disposal alternatives should be limited to cases where excessive contaminant releases can be predicted with some degree of certainty. It is worthwhile noting at this point that the proposed work is directed towards generic problems common to more than one tailings site. In this connection, the report states that R&D activities related to disposal alternatives can be divided into two types: site-specific, and those applicable to all sites, making them generic in nature. Site-specific work on disposal alternatives normally has been supported and performed by the mine operators themselves. The report recognizes that information generated as a result of these site-specific projects will be very important input to the national research program. It is emphasized in the report that the support and cooperation of mines in the proposed national program is absolutely essential. It will be necessary to gain access to selected properties and obtain permission to take required samples. There is a critical need for historical and current data that only the mines can provide. It is also recognized that the mining companies and their consultants have the technical expertise and the specialists needed for all aspects of the national program, but particularly for disposal technology. Research and development work that applies to all sites (generic) already has been the subject of broad support by federal and provincial agencies, and by industry. It amounts to approximately 20 percent of all R&D expenditures on tailings management, and falls under the headings that have already been listed, i.e., underground placement, open pit placement, deep lake placement, etc.

Cost and Scheduling

The total program has been costed out over a three-year period and funds have been allocated to the three components of the program described above. The total cost for the three years is estimated at $18.6 million, i.e., $6.9 million for modelling, $8.4 million for national measurement and $2.1 million for disposal technologies. There is also $1.2 million set aside for the operation of the National Tailings Program Office.

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It is recognized that the proposed three-year research program is not likely to provide an absolute solution to the potential problems arising from uranium tailings disposal. Satisfactory solutions to such problems may well only be found through continual effort over a span of time exceeding one decade. The main reason being, of course, the slow speed of progress of contaminants along the pathways from their source to and through the biosphere. The speed of response of the overall system to changes in the source characteristics, or pathway parameters, is such that the effect of a change may not be measurable for years. Attainment of the ultimate objective of this program is greatly complicated by the need to carefully evaluate the application of any emerging technology in economic, social and political terms. A solution is not satisfactory unless it can be shown by cost versus risk analysis that it can meet currently acceptable economic and social criteria in terms of benefits to both present and future generations.

Organization Structure

It is proposed in the Group's report that the research program be managed by a director and three assistant directors, each of the latter being responsible for the three previously described program components. They will be responsible for the detailed design and management of the program, including planning activities, selecting projects, monitoring contract research, evaluating results and preparing periodic progress reports. They will also be responsible ultimately for preparing recommendations and guidelines that would aid regulatory agencies and companies in achieving safe disposal of uranium tailings. As virtually all of the funding will be in support of contract research, there will be a requirement for technically competent scientific authorities, who may be drawn from existing government agencies and private consultants. It is also proposed that a Board of Directors, appointed by the funding agencies, will take responsibility for the overall operation of the program, including the performance of the director and his staff. It is expected that their responsibility will include the authorization of major expenditures of funds for contracts. A Technical Advisory Committee, apart from the Board of Directors, is proposed to advise the staff in developing the program and in assessing the results.

Conclusions and Recommendations

Attached.

From the Report of the National Technical Planning Group for a Research Program on Uranium Tailings

6. CONCLUSIONS AND RECOMMENDATIONS

6.1 Conclusions

As a result of its investigations, field visits and discussions, the planning group concluded that:

1. Whereas ongoing research and development work on uranium tailings in Canada has been inventoried, there has been no formal systematic attempt made to synthesize the results of measurements already made on Canadian tailings.

2. There is an inadequate understanding of the physical, chemical and biological processes that take place in the tailings and in the pathways from the tailings to the biosphere.

3. There is insufficient evidence so far, based on measurements alone as to the extent of the long-term problem in the close out of a uranium tailings basin.

4. There is a need to establish standardized measurement methodologies in order to improve the uniformity and quality of data taken at different sites across Canada.

5. Generic research and development on tailings disposal technology that applies to more than one site should be within the scope of a national program, whereas site-specific work is the purview of the mines and regulatory agencies.

6. The uranium producers' contributions to the national tailings program should be their research and development work on site-specific disposal alternatives.

The first of the above conclusions leads to the proposal to establish a national uranium tailings R&D program. The second suggests the need for a modelling program, the third and fourth for a national measurement program. The remaining conclusions refer to disposal technologies research. The conclusions collectively form the basis for a set of recommendations on uranium tailings research.

6.2 Recommendations

The planning group on uranium tailings research recommends that:

1. The federal government and the government of Ontario and Saskatchewan jointly establish a uranium tailings research and development program with a major review to be conducted at the end of three years to decide whether to continue, redirect or terminate the program, and to establish future funding and management patterns.

2. A National Tailings Program Office (NTPO) be established in Ottawa to manage the program under a program director responsible to a board of directors made up of representatives from the funding agencies, regulatory bodies and the uranium producers.

3. The tailings R and D program should consist of three sub-programs:
    - a) Modelling program
    - b) National Measurement program
    - c) Disposal Technologies program

    The modelling program, to assist in the prediction of long-term contaminant dispersion and releases, is to include the development and validation of pathway models and associated codes, the establishment of a modelling data base and the assessment of disposal alternatives.

    The measurement program, to verify models and measure contaminant levels, is to include detailed measurements on three representative tailings basins, the establishment of a national uranium waste management data base, analytical, mineralogical and other research.

    The disposal technologies program, to develop alternative long-term disposal techniques, is to focus on generic methods and build from current projects now in place.

4. Program objectives to be accomplished by the end of the third year are:
    1. A preliminary set of models and associated codes completed.
    2. A modelling data base in place.
    3. The first assessment cycle in the modelling program completed.

4. Three tailings basins fully instrumented, and at least two years of field data in the data base.

5. A national uranium waste management data base in place.

6. Major steps accomplished toward improvements in the accuracy and throughout of sample analyses.

7. Uniform standards established for field measurements.

8. A preliminary manual prepared for deriving site-specific close-out actions, based on optimization of protection guidelines.

9. Generic research on disposal technologies in place and fully interacting with modelling and measurement programs.

10. Progress made toward incorporating site-specific disposal R and D into the national program where appropriate.

5. The national program be funded in the amount of $18.3 million total over the initial three-year period, such funds to be controlled and administered by the NTPO which would subcontract to performing organizations chosen on the basis of competence and probability of meeting program objectives.

DISCUSSION OF PAPER BY V.A. HAW

A. Moorish In regard to the proposed guidlines that AECB have recently issued, was CANMET consulted?

V. Haw No. We were aware of them, saw the draft, and some comments were made.

E. Krause Can you use any of the USA data on modelling?

V. Haw The US situation in the south-west is considerably different from Canada, so I would question the validity of applicability of the data to Canada. Have looked at their data and probably we have done about as much as they have. The AECL have done considerable modelling work on high level wastes which might have application.

L. Melis Have any opinions been formed on Knight & Piesold method of tailings disposal.

V. Haw No, the committee itself made no comment. The detailed planning of the program will have to await the appointment of a Director and staff.

A. Ashbrook Comment as to there really being a problem, and some question it, but industry is supportive of the program on tailings.

V. Haw Yes, I'm glad you raised the question as we agree that industry is supportive.

I. Itzkovitch I have done a quick calculation and it would require 100 people to do the R&D. Does Canada have sufficient researchers?

V. Haw

All the work will be contracted out. Scientific authorities are required. I share your concern regarding the number of personnel required for R&D. Even the requirement to obtain the Director and Assistant Director for 3 years could be a problem. Perhaps it will be accomplished by secondments, consultants, loan from industry, but we must emphasize that 90% of the expenditure will be done outside of the government laboratories. Priorities within the various organizations will be the deciding factor on man-years allocated to such an R&D program.

Regarding the timing for the program to start, we hope that towards October-November a Director and staff will be in place so that work can start toward the end of the calendar year.

Uranium Programs at the Saskatchewan Research Council - W. Husband

1. Organization

SRC was created by an Act of legislation in 1947. Laboratory facilities and staff began operations in 1957 and today the Council has 220 people carrying out R & D in all parts of Saskatchewan. The 1980-81 budget was $6 million. 60% of this money came from contract work. Presently major activities are based in 4 buildings in Saskatoon. Numerous field stations and demonstration units are operated elsewhere in the province.

Policy is established by a board of directors, called the Council, which is chaired presently by the Minister of Industry and Commerce, the Honorable Norman Vickar. Members of Council are appointed by the provincial government. Industry, Universities and the government are represented by approximately equal numbers of members.

Originally the staff was organized on the basis of academic discipline. Divisions of Chemistry, Engineering, Geology, Industrial Services and Physics operated under a Director who reported to the Council. Each division was responsible for undertaking appropriate programs designed to develop the province through the application of science and technology.

By the middle 1970's the development of Saskatchewan's resources and the SRC's programs had reached a stage where the programs of the various divisions in SRC overlapped, and multi-discipline teams emerged. Management had to begin assigning projects to a particular division and coordinate the exchange of personnel from other divisions.

At the present time SRC is planning to change its organizational structure to replace the academic division system with a system oriented to multi-disclipine groups working in major program areas. Instead of divisions of chemistry, engineering and so on, SRC by 1982 might have a biophysical environment group, an industrial group, an energy group, etc.

2. Program

At the present time the SRC program is divided among the fields of Agriculture and Agricultural Processing, Resource Development, Industrialization, Environment and Technical Services. One can find examples of projects which relate to all fields, for example, potash mining.

In the field of Resource Development there are projects involving:
uranium ore genesis,
exploration techniques,
model borehole uranium calibration facility.

In the field of Industrialization there are projects involving:
pilot plant studies of uranium extraction processes,
removal of radium 226 from water,
potential leaching of toxic compounds from waste rock,
a radon chamber,
worker exposure in uranium mines.

In the field of Environment there are projects involving:
environmental biology in the Cluff Lake area,
radionuclide pathway studies.

In the field of Technical Services there are specialized services involving:
chemical analyses,
x-ray diffraction analysis laboratory,
electronics laboratory,
precision instrument laboratory.

Resources Research Facility

In 1977 the Saskatchewan Economic Development Corporation, SEDCO, leased 30 hectares of land from the University of Saskatchewan in order to develop a research park. The park is intended to provide a focal point for the location of applied research facilities in the province of Saskatchewan. The objective is the development of industrial and agricultural products, processes, equipment and techniques that are applicable to resource development in Saskatchewan and Canada.

Since 1977 SEDCO has constructed the 4 storey SEDCO Centre building as the first of several planned multi-tenant buildings, the park has been enlarged to 50 hectares so as to include several existing applied research facilities on the northern edge of the University's campus, and SRC has completed construction of its Resources Research Facility. This facility is made up of 3 parts, a potash pilot plant area which is leased by the Potash Corporation of Saskatchewan, an analytical chemistry laboratory, and a multi-purpose pilot plant.

Analytical Chemistry Laboratory

The sophisticated research programs operated by SRC require support from a number of service units. High technology services, not otherwise readily available in Saskatchewan, are offered by SRC to various clients on a fee-for-service basis. Among the services offered are:

chemical analyses,
x-ray diffraction analysis,
electronics design and servicing,
precision machinery,
instrument design and servicing.

Continued high demand for analytical services led to the relocation of the main analytical chemistry laboratory in the Resources Research Facility in the autumn of 1980. In 1980 this laboratory made 200,000 analyte determinations for Canadian clients including mining and chemical companies, hospitals, university research departments, government agencies, consulting firms, and city engineering departments.

The department's three Ph.D. chemists and twenty-five laboratory technologists utilize advanced instrumentation to make accurate analyte determinations and to prepare automated sample reports with analytical charge breakdowns. Sample information and analyte results are stored indefinitely on computer disk packs. A computer-based statistical processing service is available for comparing sample results. Special reports required by the Mines Pollution Branch of the Saskatchewan Department of the Environment can be automatically printed.

Major areas of expertise include water chemistry, quality control analyses, radiochemical analyses, environmental monitoring, plasma emission spectroscopy, and geochemical analyses. Neutron activation analysis began in the spring of 1981 on a SLOWPOKE II nuclear reactor.

Multi-purpose Pilot Plant

The multi-purpose pilot plant was completed late in 1980 at a cost of $825,000. The 900 square metre facility is being operated to provide clients with equipment and trained staff to test proposed processes at scales between the laboratory bench and a full-sized production operation. Work is conducted on a fee-for-service basis.

The building housing the pilot plant, the pilot plant itself, and the procedures used in operating it have been carefully designed to protect the staff and the environment from the hazards of radiation, toxic chemicals, explosive gases, explosive dust mixtures, fire, and effluents. Safety standards meet or exceed existing standards relating to waste and materials handling, personnel, plant and environmental monitoring. The entire operation has passed environmental reviews by the provincial and federal departments of Environment and is fully licensed by the AECB.

Integration of the pilot plant facility with SRC capabilities now in place will result in a substantial saving to the mining industry, ensure rapid turn around times compared with sending work out of the province, keep Saskatchewan in the forefront of uranium production technology and provide jobs in future to scientists and skilled technologists.

Other pilot plant activities in prospect that relate to the mining and energy sectors include:

- work on heavy oil recovery processes

- upgrading of Saskatchewan marl to produce lime;
- development at pilot scale of processes to make oils from straw and wood;
- investigation of liquefaction and gasification of Saskatchewan lignite.